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Riza Kusuma Uripto

A Study of New Tools to Optimize Mine Ventilation and Equipment Scheduling

Master's thesis, which has been submitted for Master of Science degree.

Espoo, 15.9.2014
Supervisor: Prof. Mikael Rinne
Advisor: Zhen Song, M.Sc

Aalto-yliopisto
Insinööritieteiden korkeakoulu

Aalto-yliopisto, PL 11000, 00076 AALTO
www.aalto.fi

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Tekijä Riza Kusuma Uripto

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Tiivistelmä

Tässä tutkimuksessa tarkasteltiin kahta ohjelmistoa, jotka toimivat työkaluina maanalaisten kaivosongelmien ratkaisemisessa. Ensimmäinen ohjelmistoista on aikataulunoptimointiohjelma, joka on kehitetty vuoden 2014 alussa. Se on osa I²Mine (Innovative Technologies and Concepts for the Intelligent Deep Mine of the Future) -hanketta Aalto-yliopistossa. Se kehitettiin optimoimaan laitteistoaikataulu maanalaisiin kaivoksiin. Tämä voi auttaa kaivoksen operatiivista osastoa hallitsemaan kaivoksen laitteistoa tehokkaasti ja saavuttamaan viikoittaisen tuotantotavoitteen.

Uutta aikataulunoptimointiohjelmaa koekäytettiin tässä tutkimuksessa, jotta sen toiminnallisuus ja kyky tuottaa kaivosaikataulu Gannt-kaaviona voidaan todentaa. Toinen ohjelman tulos on lastauspaikkojen järjestyslista kuorma-autoja lastaaville LHD (Load-Haul-Dump)-lastauskoneille. Lähtötietoina käytettiin Agnico Eaglen Kittilän maanalaisen kaivoksen viikkosuunnitelmaa. Sen jälkeen luotiin 11 skenaariota tämän viikkosuunnitelman pohjalta.

Aikatauluohjelmisto pystyy tuottamaan Gantt-kaavioita ja LHD-koneiden ja kuorma-autojen järjestyslistan perustuen joko laitteiden tai työkohteiden järjestämiseen. Molemmilla on sama kokonaisaika, koska ne optimoidaan samalla ohjelmistoalgoritmilla. Kuitenkin aikataulun optimointiohjelmistossa on vielä useita puutteellisuuksia: (1) ei voida erotella erityyppisiä materiaaleja ja lastausalueita, (2) on mahdotonta syöttää erilaisia laitteistoja samaan toimintoon, (3) lähtötietoja pitää esikäsitellä paljon, (4) hukka-aikaa, vuorovaihtoaikaa, huoltoaikaa ja räjäytysaikaa ei huomioida Gantt-taulukossa, (5) Gantt-taulukon ulkoasu ei ole huoliteltu ja (6) ohjelmistossa on joitakin ohjelmointivirheitä.

Toinen käytetty ohjelmisto on nimeltään COMSOL. Sitä käytettiin maanalaisten kaivosten ilmanvaihdon tutkimiseen. COMSOLin Computational Fluid Dynamics (CFD) moduulia käytettiin tähän tarkoitukseen. Tätä varten tehtiin yksinkertaisen maanalaisen kaivoksen malli. Malli koostuu tunnelista, jossa on neljä poikkiperää joista yksi tarvitsee räjäyttämisen jälkeistä tuuletusta. Mallinnettiin kolme skenaariota: (1) kaikki perät ovat auki; (2) kolmesta perää on suljettu LHD-ajoneuvolla; (3) kolme perää on suljettu väliseinäkankaalla (air brattice). Nämä skenaariot luotiin, jotta voitaisiin löytää käytäntö, joka maksimoi tarvittavan ilmannopeuden räjäytetyssä perässä.

Tutkimus maanalaisen kaivoksen ilmanvaihdosta osoitti, että mikään tutkituista tapauksista ei ole käytäntöön sovellettavissa, koska ne eivät tuottaneet tarpeeksi ilmannopeutta. Sen jälkeen simuloitiin, että tapa parantaa ilmavirtausta räjäytettyyn perään on asentaa toinen, ilmavirtausta ohjaava väliseinäkangas perän suuaukolle.

Avainsanat ohjelmisto, maanalainen kaivos, optimointi, aikataulutus, ilmanvaihto

Aalto University
School of Engineering

Aalto University, P.O. BOX 11000, 00076 AALTO
www.aalto.fi

Abstract of master's thesis

Author Riza Kusuma Uripto

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Abstract

This research looked into two different software, which acted as a tool for solving problems related with underground mining. The first software is a scheduling optimization software, that was developed in the beginning of 2014. It was part of the I²Mine (Innovative Technologies and Concepts for the Intelligent Deep Mine of the Future) project in Aalto University. It was developed to optimally schedule the equipment in underground mines. This could help the operation department at the mine to manage their equipment efficiently and meet the weekly plan target.

This new scheduling optimization software was trialled in this research in order to check its functionality and ability to produce the mining schedule in a form of Gantt charts. Another output of this software is the workface order list for loading dump trucks by LHD (Load-Haul-Dump vehicle). The input data for this software was from a weekly plan that was used at Agnico Eagle's Kittilä underground mine in Finland. After that, there were 11 scenarios that were created based on this weekly plan.

The scheduling optimization software was able to produce Gantt charts and the LHD-dump truck order list based on "machine set" or "workface sorting". Both have the same total time because they were optimized by using the same software algorithm. However, the scheduling optimization software still lacks several aspects, such as (1) could not differentiate different type of materials and specific dumping area, (2) it was impossible to input different types of equipment for the same activity, (3) it required a lot of input data preparation, (4) it did not incorporate break time, change shift time, maintenance time and blasting time into the Gantt chart, (5) the layout of the Gantt chart was still not perfect, and (6) there were some bugs in the program.

The second software in this research is called COMSOL software. It was used for studying underground mine ventilation. The Computational Fluid Dynamics (CFD) module of COMSOL was used for this purpose. A basic underground mine model was created based on assumption. There were a tunnel and four drifts, and one of the drifts needed post-blasting ventilation in this model. There were 3 scenarios created for this: (1) all drifts were open; (2) each 3 drifts were blocked with an LHD; (3) each 3 drifts were blocked with air brattice. These scenarios were made to find which practice could meet the minimum airflow velocity at the blasted drift.

The study of the underground mine ventilation has shown that none of the simulated scenarios were applicable for practice because they did not provide enough airflow velocity. It was then simulated that the way to improve the airflow into the blasted drift is by installing another air brattice at the drift's entrance.

Keywords software, underground mining, optimization, scheduling, ventilation

Foreword

The writing of this thesis was very important for me because it was one of the requirements to graduate from the Erasmus Mundus Mineral and Environmental Programme in general, in specifically from Aalto University and TU Delft. One of the subjects in this research, which is the scheduling optimization software, was part of the I^2Mine project at Aalto University.

Therefore, I would like to thank Mikael Rinne from Aalto University as my supervisor, who gave me the opportunity to do this thesis and have given support and guidance during the writing of this thesis. I also thank my thesis instructor and advisor, Zhen Song, who is the person who designed the scheduling optimization software, for all of his instructions and guidance during my thesis. Besides of them, I would like to thank André van Wageningen and Elen Toodu from Agnico Eagle Kittilä mine for providing the materials and support that were needed in this thesis.

Further expression of gratitude from me are to the following persons, who have given me assistance and support: Mike Buxton (TU Delft, as the co-supervisor), Juha Antikainan (Aalto University, for the discussion about mine ventilation and helping the abstract translation in Finnish), Lauri Uotinen (Aalto University, for translating the abstract from English to Finnish), Otto Hedström (Aalto University), Hans de Ruiter (TU Delft, as the coordinator of Federation of European Mineral Programs, who accepted me into this master's program), Fredy Alfonso Camargo Niño, Mateusz Janiszewski, Frans Ritala, M. Nevala, A. Heikkilä, Theo Harley Martahan, and Alexander Rosali (as my friends who have helped me with discussion and support during writing this thesis).

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Symbols

€/kWh	Euro per kilowatt hour
ε	turbulent dissipation
k	turbulent kinetic energy
kWh	kilowatts per hour
kPa	kilopascal (unit)
m/s	meter per second
m^3/s	cubic meter per second
Pa	Pascal (unit)
p	pressure (in Pascal unit)
$	US dollar
$/kWh	US dollar per kilowatt hour

List of Abbreviations

2D	two-dimensional
3D	three-dimensional
ADT	Android Developer Tools
CFD	Computational Fluid Dynamics
EUR	Euro
FEMLAB	Finite Element Method Laboratory
I^2Mine	Innovative Technologies and Concepts for the Intelligent Deep Mine of the Future
ID	Identification
IT	Information Technology
L	Level
LHD	Load-Haul-Dump
LOM	Life-Of-Mine
NAF	non-acid forming
PAF	potentially acid forming
USA	United States of America
VOD	Ventilation-On-Demand
WF	Workface

1 Introduction

1.1 Background

In these days, any types of computing machines are available to assist and solve complicated mathematical problem in a short time. However, there is still room to develop software that can optimize the scheduling of equipment, especially in underground mines. In the beginning of 2014, the I²Mine (Innovative Technologies and Concepts for the Intelligent Deep Mine of the Future) project in Aalto University had developed a new software for optimizing equipment schedule in underground mines (Song et al., 2014a). The software was in alignment to the goal of the I²Mine project, which is to improve the efficiency of underground mines in Europe (I²Mine, 2013, Song et al., 2013). This scheduling optimization software has never been used before, so there is an opportunity to try and test this new software by using data from a real underground mine. The software was created to help the operation department of underground mines, so they can optimally manage their equipment fleet to the workface on a daily basis and meet the weekly plan targets.

Ventilation system is also one of the most important thing in maintaining the operability of an underground mine. After a blasting has occurred in one of the drifts in the underground mine, the ventilation fan has to be turned on to dilute the hazardous gas and to create a tolerable mine climate around the blasted drift. There could be energy loss during this process. This loss happened because the fresh air brought by the fan went into other working areas that do not need fresh air at all. There are several ways to minimize this pressure drop, such as by using brattice cloth. However, this study would also like to see other alternatives to reduce the pressure drop, for example is by parking an LHD (Load-Haul-Dump vehicle, a special equipment that is commonly used in underground mines for loading, hauling and dumping) to block the fresh air flow to areas that do not need fresh air. COMSOL Multiphysics software was chosen for this study because it can simulate pressure drop in a 3D (three-dimensional) model of a ventilation system (COMSOL, 2013a). The COMSOL software has a feature to simulate CFD (Computational Fluid Dynamics) case by using finite element analysis. After simulating the model in COMSOL program, an economic analysis could be carried out to see the cost effectiveness of this strategy.

Both of these software programs (I²Mine scheduling optimization software and COMSOL program) are the new tools that could have a chance for solving problems related with mining engineering. Are these programs really useful for finding the solution for scheduling optimization and ventilation issue? How is the production result given by this new scheduling optimization software? How effective is it to block the airway of fresh air by using brattice cloth and LHD? This research was done in order to find answers for those questions.

1.2 Objectives

There were two purposes in this research:

a. To find out whether the scheduling optimizing software that was developed in the I²Mine project contains any bug, problems, lack of functionality or not.
b. To find out if COMSOL program is a software that could be used for investigating the pressure drop in underground mine ventilation.

1.3 Scope

The scope area of this thesis is in the scheduling optimization software and the COMSOL software. The data that were used in the scheduling optimization software were based on the data from Kittilä Mine. On the other hand, the situation of the mine that was simulated in the COMSOL program was simplified by creating a simple underground mine model based on basic assumptions. The scheduling optimization software was intended to assist scheduling in underground hard rock mines with open stoping method (not for underground coal mines).

1.4 Research problem

The problem that was formulated for this research was:

a. How is the equipment schedule made by the optimizing software in relation to the data and scenario provided by Kittilä mine?
b. What are the differences between the historical data of Kittilä (during some certain period) and the production data from the scheduling optimizing software?
c. What is the energy loss in post-blast ventilation condition, in three different scenarios (where scenario 1 is "all road access are open to airway"; scenario 2: "access to working drift without blasting are blocked using curtains"; scenario 3: "access to working drift without blasting are partially blocked by a mining equipment") in the COMSOL software?
d. How cost effective is it to block the airway by using a parked LHD?

1.5 Structure of the thesis

The first chapter is the introduction part which explains the background behind this thesis. The second chapter is *Literature Study and Theory* which is a brief explanation about studies that were done before related with this thesis, the theories that underlie the operating nature of the software and explanation about both software interfaces. The third chapter is the *Methodology* chapter. It is about the steps that were done during the research in order to obtain the results. The fourth chapter is the *Result and Discussion* chapter. The final chapter is the *Conclusion and Suggestion* chapter which concludes this writing.

2 Literature Study and Theory

2.1 Underground mine ventilation

Ventilation is required in underground mines to support the operation. All personnel and equipment need oxygen to operate, thus fresh air is required in the underground. Besides of that, the dust in the opening must also be diluted, so workers could work with clear visibility and their lungs would be safe from respiratory complications. The fresh air is usually brought through the intake shaft or decline, while the exhaust air is transported out from the underground mine through the exhaust shaft.

The amount of workers and equipment inside the underground mine would determine the minimum requirement of fresh air into the mine. Usually, there are mining regulations in every country to determine the minimum level of oxygen and the maximum level of hazardous and flammable gas (such as methane, carbon monoxide, carbon dioxide, and hydrogen sulphide).

To ensure the atmosphere of the underground mine would fulfil these requirements, an underground mine would install a ventilation system. This system would consist of the main ventilation fan (either as an intake fan or an exhaust fan), auxiliary fan, air ducts, and air regulators. Air regulator is needed to control the direction of the airflow in the mine. This could be in the form of brattice cloth (see Figure 1), air door, air curtain or air bridge. The main purpose of using those fans and regulators is to ensure the required airflow would flow properly in the underground mine. An example of this illustration could be seen in Figure 2.

Figure 1. A brattice cloth installed in an underground mine to block the airflow (Minvent Solutions, 2014).

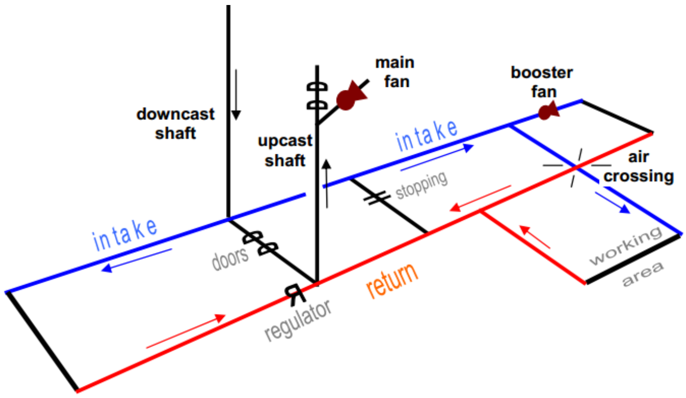


Figure 2. Illustration of an underground mine with the ventilation system (McPherson, 1993).

Several methods to improve ventilation could be done by doing proper ventilation, planning the exact auxiliary ventilation, and conducting day-to-day examination of the mine ventilation (Pritchard, 2010). Another way to optimize ventilation is by actively controlling the ventilation by using the "Ventilation-On-Demand" (VOD) technology (Gundersen et al., 2005). This active control uses technology (such as environmental sensors, Real Time Location System, operating parameters and advanced communication infrastructure) to intelligently identify areas in the mine that need more airflow and which area that does not need airflow at all. There are several examples where VOD could optimize the underground mine ventilation system. At Vale's Coleman mine in Sudbury, Canada, the trial implementation of VOD at only three levels of their underground mine for around 4 months was expected to save $153,000 per year (Allen and Tran, 2011). While at Barrick's Goldstrike underground mine in Nevada, USA, a daily saving of about $5,000 in energy costs was looked forward to as the result of implementing VOD and energy management system (Meyer, 2009).

2.2 Computational Fluid Dynamics (CFD) in mine ventilation

2.2.1 The usage of CFD

CFD is defined as the application of an amount of numerical methods in order to gain the solution for problems related with fluid dynamics and heat transfer (Zikanov, 2010). By using numerical methods, CFD is just a way of how to solve problems in fluid dynamics and heat transfer.

CFD has been used since the 1950's for a lot of research in the academia and the industry. It gained more popularity together with the development of computers. It has been widely used for evaluating aerodynamics, acoustic, or thermal coupling. Therefore, it had been better known in industries such as aerospace, automotive, and chemical processing. There has been a lot of research in the mining academia and the mining industry by using CFD because it is simpler than conducting a large scale experiment at the laboratory or at the field (Siddique et al., 2005, Ren and Balusu, 2010).

The validity of CFD code to simulate underground mine ventilation had been proven by a research at University of Kentucky (Wala et al., 2008). They compared the results given from the CFD with data from an experiment in their testing facility. They were looking for the concentration of methane gas at the face of a drift and verified similar results between the simulation in CFD and the experiment in their laboratory.

CFD was used in a research at Southern Illinois University Carbondale to find out the airflow around a continuous miner that is doing a right turn cut in an underground drift (Kollipara et al., 2012). Their study showed how CFD could analyse different airflows around the continuous miner when it is performing under different scenarios. They could show how leakage in the line curtain could reduce the fresh air volume to the continuous miner and low velocity of the air when the continuous miner is performing the right turn cut.

A study by Diego et al. (2011) was looking into the applicability of CFD to find out pressure loss of ventilation for underground works. They collected data from mining and tunnelling projects in Spain, and compared those data with calculation results by using CFD. They concluded that CFD is adequate and sufficient to find pressure loss in the ventilation system.

2.2.2 Cost for mine ventilation

The mine ventilation system could affect the required capital cost and operating cost of the mine project. According to an article published in the website of CANMET by Brière (2012), it is mentioned that around 40% of the total cost of power consumption is for operating mine ventilation. To find out the total cost for ventilation, one could inquire quotation from companies that produces equipment for ventilation, such as fan producers, ventilation tubing companies and brattice producers. Another way to estimate the ventilation cost would be by using information from existing database. A good example of this kind of database is provided by InfoMine USA (2010). For example, an estimation of ventilation cost for a cut-and-fill-mechanized-shaft underground mine made by InfoMine USA can be seen in Appendix 2. InfoMine USA cost book is very comprehensive because one could find the estimated capital cost to purchase a fan ventilation, fan tubing, brattice cloth. The operational cost could be estimated by finding out how much electrical power (in kWh) is needed to operate the ventilation system annually. The number of operating days per year and operating hour per day will be taken into account when calculating the annual power consumption. After knowing the amount of electrical power, this will be multiplied by the electrical cost rate for the mine (this can be either in $/kWh or €/kWh) to obtain the annual cost for operating a ventilation fan.

2.2.3 The principal theory of CFD

The governing equations of fluid dynamics are the main fundamental bases that build CFD. Those equations illustrate the conservation laws of physics in mathematical explanation. The physical laws related with CFD are:

a. The mass is conserved for fluid (conservation of mass/continuity).

b. Newton's second law, where the amount of change of momentum equals the total forces acting on the fluid (conservation of momentum).

c. First law of thermodynamics, where the amount of change in energy equals the total of heat rate in addition to and the amount of work done on the fluid (conservation of energy).

2.2.4 COMSOL software program

COMSOL Multiphysics software has a capability of running different type of simulation. It has 12 packages inside the software that the COMSOL developers called "modules". For example, there are modules about geomechanics, acoustic, fatigue and CFD. The CFD module is the one that was used during this research. Several examples of simulation that this software could do according to the demonstration file are study of airflow in an airlift loop reactor, two-phase flow modelling of a dense suspension, and contaminant-removal from wastewater in a secondary clarifier (COMSOL, 2014).

The software is only a tool and it is the user of the software who has to define what kind of simulation they wanted to use in this software. The software is widely used in lots of kind of industries due to those available modules. In the CFD module, the users could select different kind of phase of fluid, such as single phase or multi-phase. The basic of the CFD module is as described in the two previous sub-chapters.

Certain boundaries of the model had to be setup in the software before creating the meshes. The processing of CFD is a numerical method system, thus these meshes are required to simulate how each element would interact to each other.

When evaluating the results from simulating the model in CFD, it has be made sure that the wall lift-off in viscous unit has to be not greater than 11.06 (Frei, 2013). Wall lift-off in viscous unit is an indication of whether the results obtained from the simulation is accurate enough or not (Grotjans and Menter, 1998, van Schijndel, 2012). If this indication has a value greater than 11.06, the references suggest the users to modify the mesh element size. The modification of the mesh element size could be done from that current mesh size ("coarse" size for example) to become a "finer" (or smaller) mesh size. This modification has to be done in order to obtain better modelling results with higher accuracy.

2.2.5 The ventilation model in this research

There were three certain scenarios of the underground mine that were simulated in the COMSOL program. These three scenarios are:

a. Scenario 1: All non-blasted drifts (these drifts have workface without any blasting) have free and open airway (not blocked).

b. Scenario 2: All non-blasted drifts are blocked with parked LHD equipment.

c. Scenario 3: All non-blasted drifts are blocked with brattice cloth.

The illustration of these scenarios can be seen in Figure 3. It was assumed that there was a blasting only at the tunnel end of Drift 3, so Drift 3 should get the highest priority for fresh air ventilation. These scenarios were generated in the COMSOL program and the result from each scenario could be compared in order to give more idea of what would happen during the simulation.

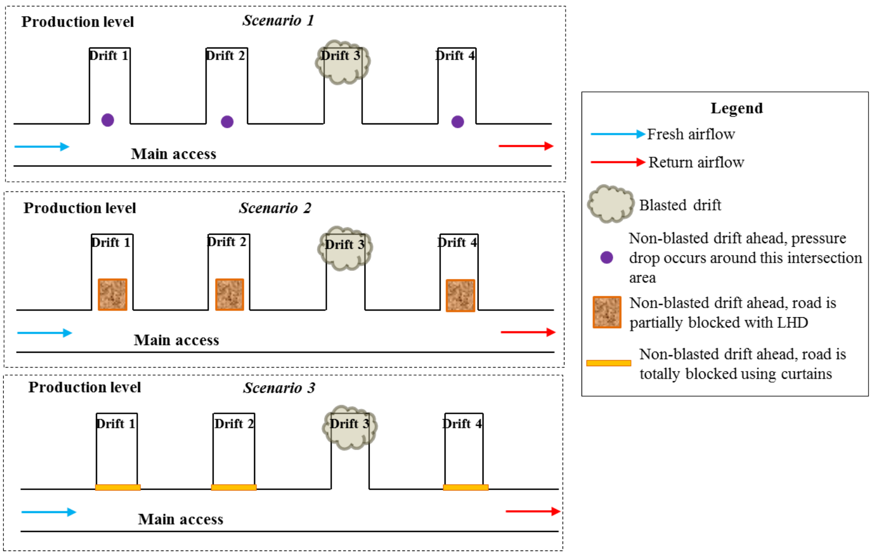


Figure 3. The three scenarios that were simulated in COMSOL software (the sketch is in top view).

At the Pyhäsalmi underground mine in Central Finland, there is a project to develop the underground tunnels as a physics research facility. According to the feasibility study document of that project, it is required to have an airflow rate in the ventilation of about 1 m/s to dilute the dusts and gases from blasting (Rockplan, 2010). Therefore, referring to the ventilation requirements at Pyhäsalmi mine, it was expected for the simulation of all scenarios in this research to also obtain a minimum velocity magnitude of 1 m/s at all drifts.

2.3 Equipment scheduling optimization by software

2.3.1 Underground mining phases and scheduling

There are several phases that usually exist before developing an underground mine. The phases are: exploration, preliminary study, development and production and mine closure. The exploration phase is for defining how the shape, quantity and quality of the ore deposit look like. After that, several study has to be commenced before deciding a go or no-go to mine this deposit. This preliminary study would produce the mine scheduling, such as how many tons of ore would be mined per year and per month. Then, the next phases such as development, production, and mine closure are relatively a straight forward process. Those phases are just more about executing the long-term mine plane that had been already made before.

When the mine is under the developing and operating stage, this is where the problem usually comes because when operating the mine, there are a lot of various events. Usually, the most unpredicted event is the unpredictable event of equipment breakdown. The closest thing a maintenance engineer could do to maintain and manage the equipment is by using statistical approach of the historical database. Deciding an equipment to go into maintenance would mean taking the equipment out from the operational fleet into the workshop. This would mean a loss of opportunity for production. This loss would mean a lot, especially when the maintenance was an unexpected maintenance because it would make the actual production of that day to be less than planned before in the original mining plan.

Real time monitoring of the equipment have been developing for the last decades thanks to the advancement in the IT industry. This real time monitoring has helped the operation department to find out where the location and condition of their equipment in the mine. The supervisor could ask the central control office or the dispatch controller about the equipment availability because the controller could detect the equipment status.

However, this dispatch system has still somehow did not incorporate the possibility to assist mining supervisor about to which drift should they send their mining equipment in an optimized manner. This optimized decision process was the trial that had been developed in Aalto University (Song et al., 2014b). The details of this software can be seen in Sub-chapter 2.3.2.

The scheduling optimization that has been done in most underground mines is for scheduling sequence of stope production and maximizing the net present value (NPV). An example of this is by using integer programming model of an underground gold deposit done by Little et al. (2013), which produced higher NPV rather than the isolated approach. Examples of software that are available on the market for production scheduling in underground mines are Desvik.Scheduler and MineRP EPS.

2.3.2 Algorithm of the scheduling optimization software

The algorithm of this software was explained in a paper written by Song et al. (2014b). The algorithm works in a way where the distance of each drift is constructed as a matrix. The statistical machine data (such as moving rate and operating rate) and the external constraints (such as dump area capacity, material amount at the workface, distance from workface to dumping area, machine initial location) are also included. They are also in the form of matrixes. All of the workface that are located at the same level are clustered together. After that, the time needed for operating in a serial and cyclic manner were calculated together with those external constraints. The clustered levels were sequenced and then this could be used for seeking the timestamps and scheduling the machines at each workface (Song et al., 2013). The factors related with the algorithm of scheduling the equipment and the workface can be seen in Table 1.

Table 1. List of key factors which are important in the scheduling optimization software.

Key factors in the equipment scheduling optimization software		
• The distance from one working face (drift) to another working face	• The distance from the loading point to the dumping point	• The amount of material that could be excavated from a workface
• Equipment productivity	• Dump capacity	• Equipment type
• The required time to travel to another location	• The required time for an equipment to operate at an workface	

The scheduling optimization software uses Java programming language as the underlying platform. Several rules had to be followed in the software when compiling the input data. This is due to the matrix format that is used in the algorithm. The input data that had to be inputted into the software can be seen as in the list below:

a. Data about *the dumping area*. The details for the dumping area are the name of the dump site and the capacity of the dump site.

b. The *distance between the dumping area and the loading area* in meters. This would help the program to determine to which dump area would be the best option to go.

c. Data about the *initial location of the mining equipment*. This shows where the equipment is located in the underground mine, and to which workface would be the most efficient way to go.

d. Data about the *machine operating info*, where there are two specific rate used. These rates are the operating rate and the moving rate. Operating rate is the rate of the equipment to perform its purpose at one face, in meter per hour. For example, a drilling machine that has an operating rate of 2 meter/hour means that this equipment could drill a hole of two meters in one hour. Moving rate is the rate of an equipment to move from one location to another location, stated as meter/hour in the program. This is the same thing as the speed of the equipment to travel from one location to another, which is usually stated as km/hour (metric unit) or mile/hour (imperial unit)

e. Data about *the machine set*. Machine set is more about grouping and clustering all of the machines in the mine into different fleets.

f. Data about *the truck*. The required data are the truck speed (in meter/hour), loading time of the truck (in hour), and the truck payload (in ton).

g. Data about *the workface dependency*. Workface dependency means how dependent is one workface to another workface prior to be developed or exploited. This has to do with the long term mining plan, geological considerations, and economic considerations. For example, if there are two adjacent workface, named as workface A and workface B respectively, workface B for instance cannot be developed immediately before workface A has already been developed due to geotechnical reasons. But in some situations, a workface can be independent and not depending on another workface.

h. Data about *the travel distance between one workface to another workface*. This distance is stated in meter. This data is needed to give an indication for the equipment to travel from one workface to another workface.

i. Data about *the amount of material that has to be excavated from a workface*. This will indicate the needed truck and loaders to excavate the blasted materials from a workface.

j. Data of *the workface priority*. Every workface has its own level of priority; the workface with higher priority will be prioritised to be developed first instead of the lower level workface. Thus, the equipment will go to the workface will higher priority first.

k. Data about *the workface workload*. In the scheduling optimization program, the workface workload is stated in meters. Workface workload in meters means how many burden the equipment has to remove before moving on to the next assignment/workface. For example, if there is a workface with a workload of 4.5 meters, then this a drilling machine with an operating rate of 1.57 meter/hour would have to operate for 2.87 hours at that workface.

All of these data were inputted into the optimization program after they were all available. The software could only recognize all of these data as a group of several matrixes. These matrixes were then computed based on permutation. From these permutations and calculations, the algorithm will find its way for the most optimized schedule available.

2.3.3 User interface of the software

The scheduling optimization software was developed on a developer program that is usually used for developing Android applications. This developer program is called ADT (Android Developer Tools v22.0.1-685705) and it acts as a plugin on a software called Eclipse. The programming language that is usually used for this developing program is the Java programming language (Android, 2014).

All of the input data that were going to be processed in the scheduling optimization software had to be sorted and divided into specific text files (*.txt). The list of these text files can be seen in Table 2 (page 21).

Table 2. List of the text files which are required for the scheduling optimization software.

Text files for inputting the data		
• Dump_Site_Capacity.txt	• Machine_Set.txt	• Workface_Mineral_Capacity.txt
• Dump_Workface_Distance.txt	• Truck_Info.txt	• Workface_Workload.txt
• Machine_Initial_Location.txt	• Workface_Dependency.txt	
• Machine_Operating_Info.txt	• Workface_Distance.txt	

The software window consists of several menu tabs such as "Read in basic files", "Perform operations" and "LHD". The "Read in basic files" tab area is where all of the input data are loaded into the scheduling optimization software. The "Perform operations" tab area is where the user can select on how the scheduling would be based on. There are four types of basis for the scheduling (Song et al., 2014a):

a. "Schedule by workface priority": The scheduling is based upon the priority level of the workface. There are three levels: level 1, level 2, and level 3. Level 1 has higher priority than level 2, and level 2 has higher priority than level 3. This is inputted in the "Workface_Priority.txt".

b. "Schedule by sharing machines": The scheduling is based on the machine availability and the amount of equipment fleet (named as "machine set" in the software). Users can choose one fleet, two fleets or three fleets as the consideration for the scheduling.

c. "Schedule by workface dependency": The scheduling is based on the dependency of one workface to another workface. If workface 2 is dependent on workface 1, then workface 1 will be given higher priority by the program to be in the schedule first.

d. "Schedule after sorting the workface": The scheduling is based on the distances for every workface that could be grouped together, thus creating a more efficient scheduling for the equipment without too much traveling time.

The interface of this "Read in basic files" and "Perform operations" tab can be seen in Figure 4. The green checkmarks indicate that the required text file has already been uploaded into the program.

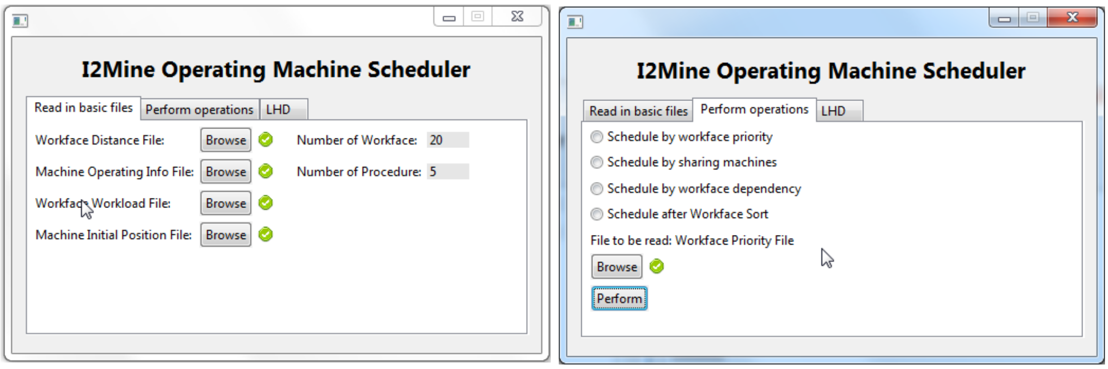


Figure 4. Interface of the "Perform operations" tab and the "Read in basic files" tab, where the required files are already selected (indicated by the green checkmark).

After the "Perform" button has been clicked, the program will run the algorithm to produce the optimized schedule. A new window will appear after this process has finished. It is Gantt chart of the schedule and it is based on the selected type of scheduling. This can be seen in Figure 5.

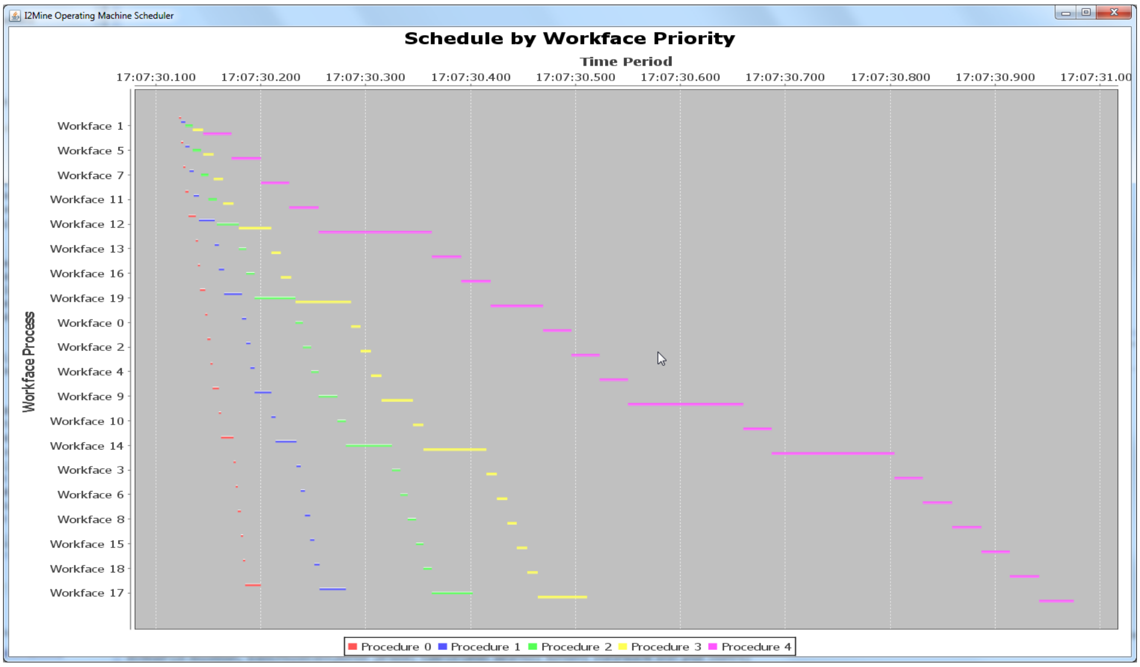


Figure 5. An example of the Gantt chart based on "Workface Priority", produced by the scheduling optimization software.

The final tab in this scheduling optimization interface is the "LHD" tab menu. This feature is for creating a list of where the dump trucks and the loading equipment (LHD) should be sent for operation in a fashioned order. The program processed the information of the distance between all of the workface and the dumping location, the dumping area capacity, the truck specification, and the amount of material that has to be removed from each workface. The interface of this menu can be seen in Figure 6.

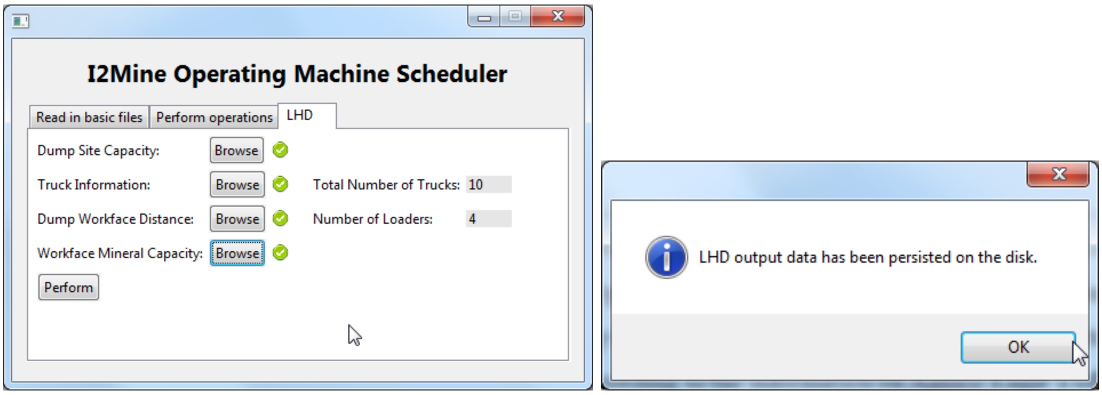


Figure 6. The appearance of "LHD" tab menu in the scheduling optimization software (left) and the pop-up message after clicking the "Perform" button (right).

The result after running the "LHD" feature can be seen in Figure 7. This result appears in the console window of the Eclipse software. It shows by order, what is the location of the workface (identified by the workface identification number), the starting time and ending time of the loading and hauling activity, the duration of the activity and the amount of truck needed for the loading and hauling activity.

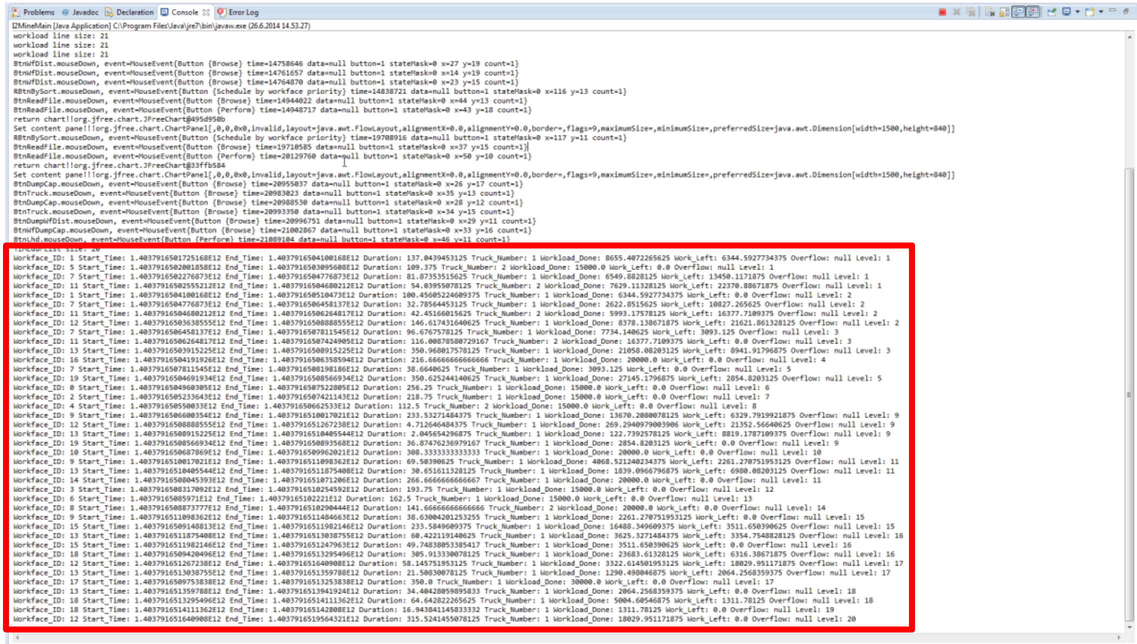


Figure 7. Results from processing the input data of loaders and truck in the "LHD" tab menu, shown in the red box area.

2.3.4 The Kittilä mine

Kittilä mine is an underground gold mine located at Lapland, Finland. This mine is owned by a Canadian company called Agnico Eagle Mine Limited. It started its mine production on 1 May 2009. According to the information on Agnico Eagle's official website (Agnico Eagle, 2014b), this mine has proven reserves of 4.7 million ounces and probable reserves of 32 million tonnes at 4.6 g/t gold. The mine produces around 3,000 tonne of ore per day and it is targeting to produce 4,300 kilograms of gold in 2014. After this, the average production will be roughly around 4,700 kilograms of gold per year in 2015 to 2016.

Originally, Kittilä mine started as two open pits which are called Suuri and Roura. Then, they had developed the underground mine in October 2010. The cross section of the mine could be seen in Figure 8. All of the pits had already been mined out in November 2012. Therefore, only the underground mining operation exists.

The mining method that they use in the underground mine is by developing open stoping. Open stoping is the space in the ore body where ore is divided into sections and the ore extraction takes place in sequences for maintaining the ground stability. When a stope has already been mined out and void, they would backfill this void with cemented backfill or paste backfill to retain the ground stability of the mine. This method proceeds for the next sections as in the planned sequence.

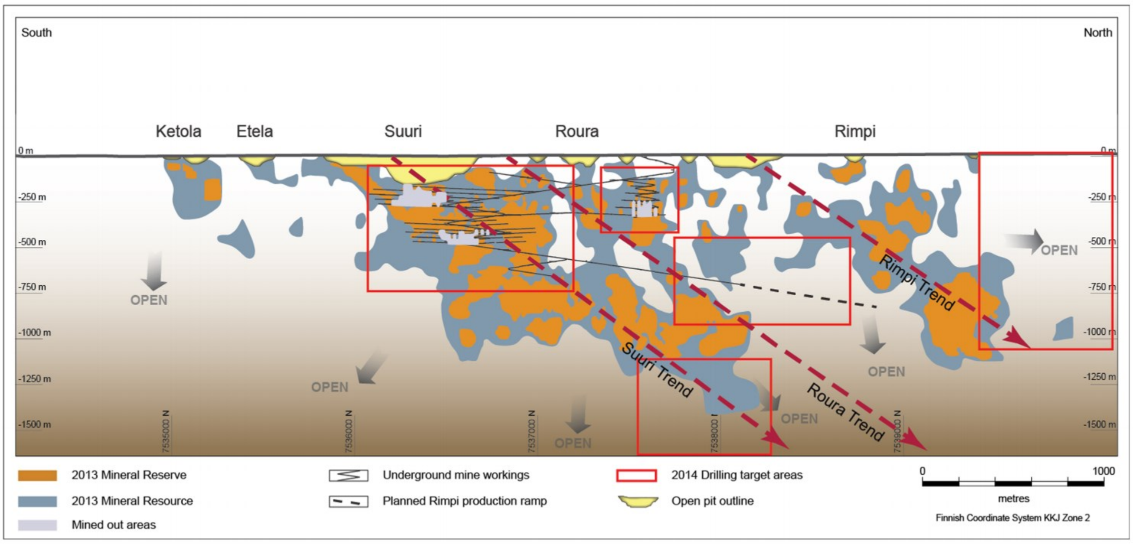


Figure 8. Cross section of Kittilä underground mine (Agnico Eagle, 2014a).

The broken ore is hauled by truck to the surface and dumped at the ore stockpile. This stockpile will feed the main crusher which is next to the processing plant. The ore is processed by grinding, flotation, pressure oxidation and carbon-in-leach circuits. Output of the leach circuit is sent to the electrowinning process to recover the solution and remove the carbon. It will then be smelted in a furnace and moulded into doré bars. The gold recovery from the mining and mineral processing during the life of mine is targeted to be more than 89%.

Most of the drifting activity is currently happening in the Suuri deposit, ranging from level 200 to level 550. This is around 200 to 550 meters below the ground surface. The deepest level that they are working so far is at around level 650 and they are continuing this to follow the Suuri Trend. The main underground workshop is located at level 350 (Haga, 2013). In Kittilä mine, they have two types of equipment fleet. They are the development fleet and the production fleet. The development fleet is used for constructing tunnels and opening in the underground, such as ramp and drift. Every time the development fleet operates to development the tunnel, the tunnel will advance for around 4.7 meters. The list of equipment in the development fleet of Kittilä mine can be seen in Table 3.

The production fleet is the fleet that is used to construct the open stope, in order to produce ore from the ore body. This fleet has specific drilling equipment (production drillrig) that have the ability to create fan drilling for fan blasting in the ore body. Besides of that, it also has loader equipment to load the trucks with blasted material. The list of the equipment in the production fleet of Kittilä mine can be seen in Table 4.

Table 3. The development fleet which is used at Kittilä mine.

Development fleet	Machine model	Equipment amount
Scaling	Normet Scamec	2
Cleaning (excavator+loader combined)	Mecalec and CAT 930	3
Shotcreter	Normet Spraymec 1050	2
Support - Bolting	Atlas Copco Boltec LC	3
Drilling	Atlas Copco Rocket Boomer E2 C22	2
Charging	Normet Charmec 1610	2
Services - Water truck for dust spraying	Scania	1

Table 4. The production fleet which is used at Kittilä mine.

Production fleet	Machine model	Equipment amount
Truck	Scania R 560	10
Loader	Sandvik LHD 517	3
Loader with hydraulic hammer	Sandvik LHD 517	1
Fueling equipment	Normet Multimec 1000	1
Road maintenance - grader	CAT 14H	1
Production drillrig	Atlas Copco SIMBA M6 C	1
Production drillrig	Atlas Copco SOLO DL430-7C	2
Production charging	Scania + Forcit	1

Although the loader (LHD) and trucks are listed in the production fleet, they are divided evenly for both the development and production activity. Therefore, usually there is one loader with 3 to 4 trucks doing load and haul activity for the development front; and another loader with 4 to 5 trucks doing load and haul activity for the production front.

Figure 9 (page 27) depicts the layout of the weekly plan that is usually used in Kittilä mine. It is specifically of the time period between 4 September 2013 and 10 September 2013. The upper part of the weekly plan shows the priorities for which drifts have to be mined. The rest of the parts in the weekly plan shows the information such as the remaining blasted material in the drifts, remaining meters that have to be drilled, and certain attention that have to be given for certain drifts (for example the risk of roof collapse and unavailable road access).

In order for Kittilä to create the weekly plan, they first start from the longest horizon of the mine plan, which is the Life-Of-Mine (LOM) plan. From the LOM plan, the 18 months plan is derived. After that, the monthly plan is created according to the 18 months plan, and finally the weekly plan is made out from the monthly plan. The weekly plan is implemented at the field by communicating the plan to the head and supervisors of the operation department. These supervisors usually use their own experience and personal judgement when encountering problems of implementing the weekly plan. This kind of basis could often become biased and subjective. This is where the scheduling optimization software could assist them, by making an optimized daily plan and helping them to make the best decision, thus fulfilling the weekly plan targets.

VIIKKOSUUNNITELMA/WEEKLY PLAN

4.9.2013 ke 10.9.2013

		Katkot	Metriä	Kuun tavoitetta jäljellä (katkoa)	SEURANTA	KOMMENTIT
1	VT-TUT	2	9.0	16		VEDENPOISTO
2	VT1	2	9.0	9+5		
3	200P134	2	9.0	6		Porapultit asennettava ennen louhinnan aloittamista
4	200P140	2	9.0	4+3		
5	225P122	1	4.5	3+3		
6	225TP2	1	4.5	6		
7	250P137	1	4.5			Yläpuolen louhosta varten
8	250P122N	1	4.5	1		Läpi puhkaisu
9	250PP120S	1	4.5	3		HUOM! UUSI PORAUSKAAVIO
10	375P153	VIUHKA	3.0	3		
11	375P154	1	4.5	1		
12	375P155	VIUHKA	3.0	3		
13	375P159	VIUHKA	3.0			Lastauspaikka tähän risteykseen.
14	375TP1	1	4.5	3		
15	375TP2	1	4.5	4		
16	400P135	VIUHKA	3.0	3		
17	400P136	1	4.5	1		Voi ottaa viuhkan ja kaksi katkoa, jonka jälkeen STOP.
18	400P154	1	4.5	2		
19	400P158	1	4.5	4		
20	400P159	1	4.5	4		
21	400TP1	1	4.5	4		
22	400TP2	1	4.5	4		
23	425P139	VIUHKA	3.0	3		
24	425P140	1	4.5	1		GEO STOP
25	425P141	1	4.5	1		GEO STOP
26	425P145	1	4.5	2		
27	425P149	1	4.5	3+3		
28	425TP2	1	4.5	5		
29	450PP137S	1	4.5	3		HUOM! UUSI PORAUSKAAVIO
30	450PP150S	1	4.5	3		HUOM! UUSI PORAUSKAAVIO
31	450TP1	1	4.5	4		
32	475P131	1	4.5	3		
33	475PP137S	1	4.5	20		GEO STOP. HUOM! UUSI PORAUSKAAVIO
34	475TP1	1	4.5	3		
35	500TP1	1	4.5	5		
36						
		34	168.0	121.0		

TUOTANTO

		PORAMETREJÄ JÄLJELLÄ	IRROITTAMATTA t	IRTOJA LASTAAMATTA	SEURAAVAT RÄJÄYTYKSET	
PORATUT	R350L217 AK VAIHE1 LEIKKAUKSET 1-7	0	0	7,400		
SIMBA	R350L217 VAIHE 2	1,400	21,100			
SOLO1	R270L216 yk vaihe1	633	5,600			
	450PL143	690	6,200			
SOLO2	475L147 nousu					Alapuolen louhos täyttämättä
	450PL142 nousu					
	R310PIP2-R350PIP2					
L8	225L130					

LOUHOSTÄYTTÖ

	Jäljellä		KESKIVIIKKO	TORSTAI	PERJANTAI	LAUANTAI	SUNNUNTAI	MAANANTAI	TIISTAI	Yht	
225L129	?									0	
R350L215 raakku täyttö	8,900	15000 t kiviä								0	
500L147B	1,500									0	
300PL125	1,600									0	PASTALINJAN RAKENNUS JA TULPAN TEKO
250L140	10,000									0	
475L151	2,800									0	
500L145	1,900										

VAIJERIPULTTAUS

	KESKIVIIKKO	TORSTAI	PERJANTAI	LAUANTAI	SUNNUNTAI	MAANANTAI	TIISTAI
350PESU2							
R675PIP							
450PL142+143							
475-TASON RISTEYS ALUEET P150-148							
475L147							
R270L216							
250L137							

KAIRAUS

		KESKIVIIKKO	TORSTAI	PERJANTAI	LAUANTAI	SUNNUNTAI	MAANANTAI	TIISTAI
NTK	425TP1	X	X	X				
	R675RIP				x	x	x	x

TEHTÄVÄT

		KESKIVIIKKO	TORSTAI	PERJANTAI	LAUANTAI	SUNNUNTAI	MAANANTAI	TIISTAI
1	VEDENPOISTO/HALLINTAAN SAANTI							
2	R350 PIP Bergteametille pohjan valu							
3	225 pumppaamon ritilän umpeenvalu.							
4	200Lip1 ja2 pohjien valut							
5	VT1 suuaukon RB							
6	425Lip pohjan valu							
7	540 valmistelu Bergteametille							
8	225 pumppauksen järjestely (AJH-INFRA)							
9	450RIP2 tuuliseinä							
10	475RIP2 tuuliseinä							
11	540 rintin reikien poraus							

Figure 9. The layout of the weekly plan that is usually used in Kittilä mine (Agnico Eagle, 2013).

3 Methodology

3.1 Research step for simulating the ventilation in COMSOL

This sub-chapter explains the procedures that were conducted during the simulation by using COMSOL program. In general, there were six steps that had to be done in order to complete a model simulation in COMSOL program. According to the manual (COMSOL, 2013b), those steps are:

a. Creating the geometry.

b. Defining the material.

c. Defining the physics of the model.

d. Constructing the mesh of the model.

e. Running the study and processing of the model.

f. Plotting the results as plot groups.

3.1.1 Creating the underground mine geometry

In the starting phase of the COMSOL software, the environment of the simulation had to be defined first. This was done in the program through: Model Wizard → 3D Space Dimension → Stationary study.

It was set to the standard k-ε model, with turbulent flow and single phase. The standard k-ε interface is usually used for simulating single-phase flows at high Reynolds number. The k is the turbulent kinetic energy, where this is the energy of the turbulence in the fluid. The ε is the turbulent dissipation which expresses the dissipation rate of the turbulent kinetic energy. The stationary study was chosen because it was assumed that the field variables do not change over time.

In the "Global Definitions" settings, the initial velocity was defined as 4 m/s in the parameters window. After this parameter was set, it continued with the construction of the geometry of the underground mine model in the graphics window.

There were several assumptions that were made for the geometry of the underground mine, which are the following:

a. For the main access tunnel: width 6 meters, height of side wall 5.5 meters, height to the peak of the curve: 6.85 meters, length 73.5 meters.

b. For the drifts: width 6 meters, height of side wall 5.5 meters, height to the peak of the curve: 6.85 meters, height 5.5 meters, length 40 meters.

c. Distance between each drift: 8.5 meters.

d. Distance from the inlet and outlet of the main access tunnel to the closest drift: 12 meters.

By creating the underground model as detailed in the previous paragraph, the surface wall of the whole tunnel becomes a simple smooth wall. This is totally different to most actual tunnel wall in underground mines because the latter are usually rough and blocky. An illustration of these assumptions can be seen in Figure 10. The model in Figure 10 was specifically used for Scenario 1. Figure 11 shows the tunnel profile of the drift and the main access tunnel.

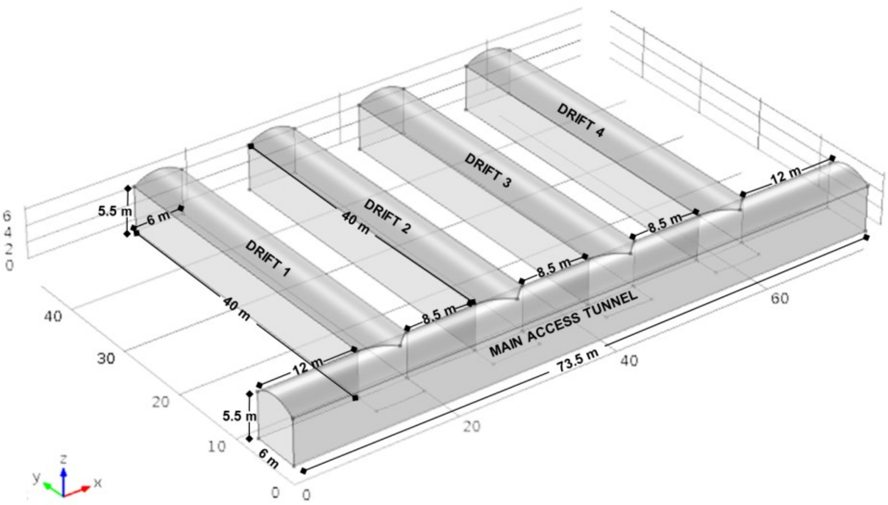


Figure 10. Geometry of the underground mine model in COMSOL for Scenario 1.

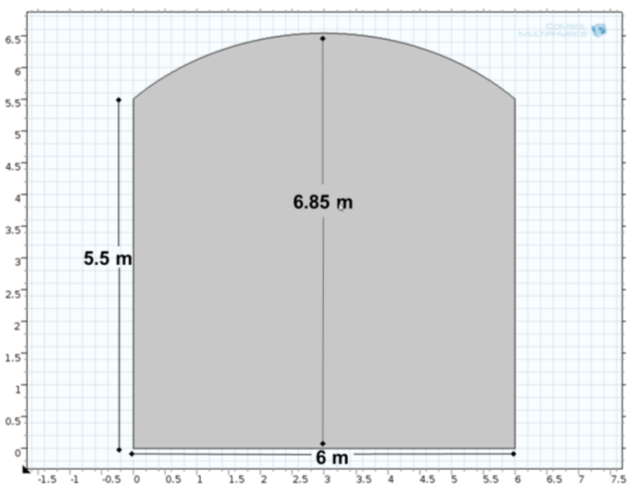


Figure 11. Tunnel profile of the drift and main access tunnel.

The geometry model of Scenario 2 was based on Scenario 1 but there are three drifts blocked with an LHD (see Figure 12). The LHD's are located in Drift 1, Drift 2 and Drift 4.

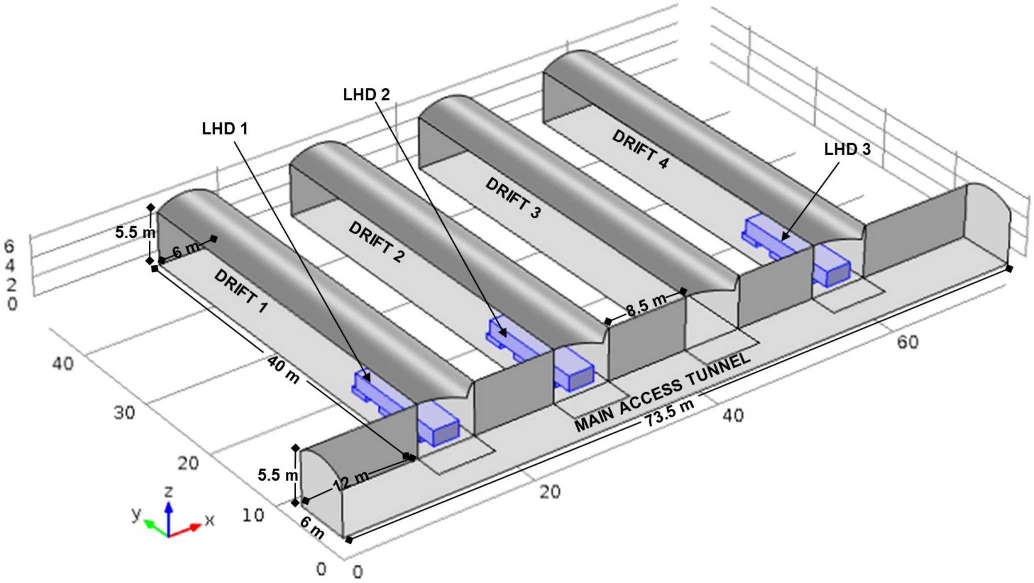
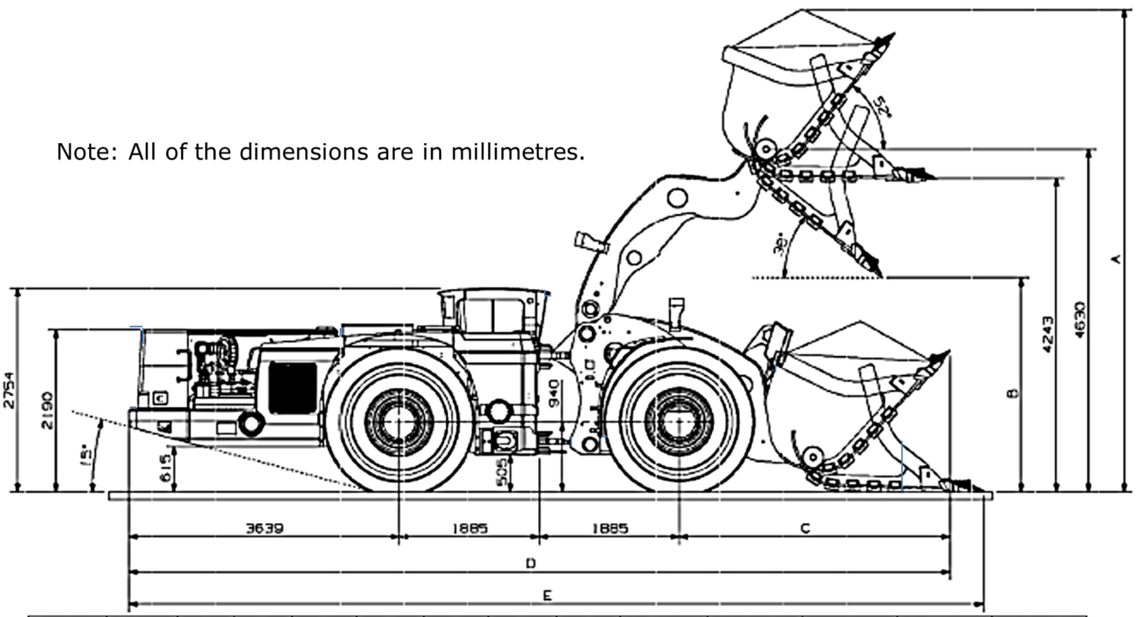


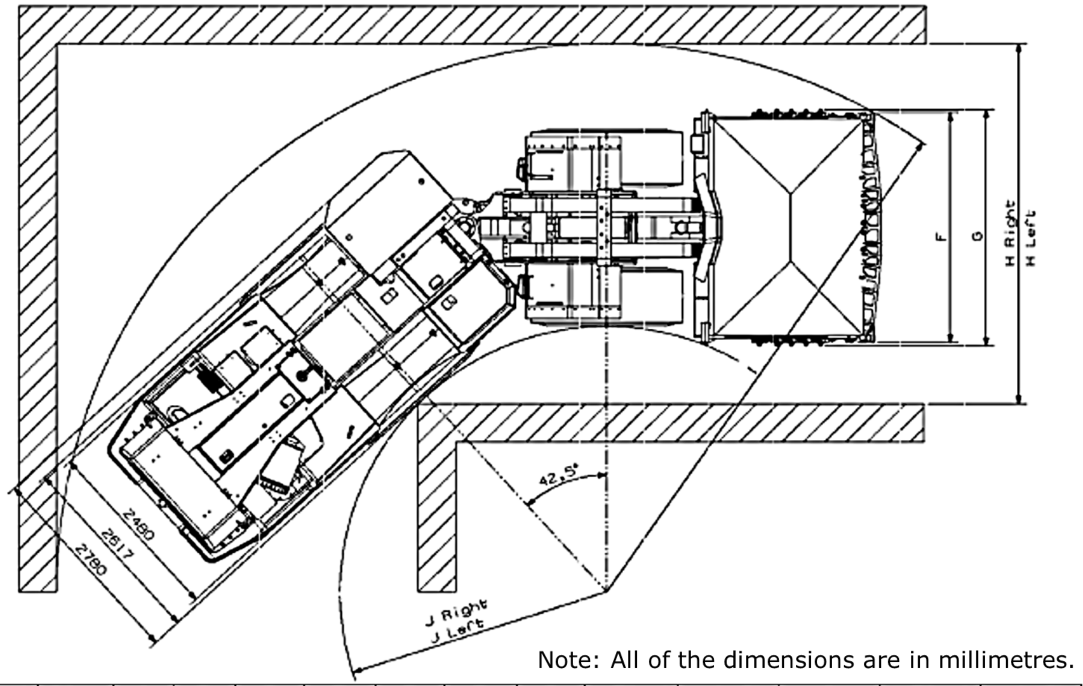
Figure 12. Geometry of the underground mine model in COMSOL for Scenario 2.

Figure 13 and Figure 14 shows the illustration of the Sandvik LH517 with the detailed dimensions of the LHD (Sandvik, 2010). This particular type of LHD was used as the basis for modelling the LHD in Scenario 2. The illustration of the LHD model that was created can be seen in Figure 15.



Bucket	Size	A	B	C	D	E	F	G	H Right	H Left	I	J Right	J Left
Shark	7.0m3	6522	2896	3646	11055	11506	3066	3148	4798	4841	R7297	R3534	R3474

Figure 13. The dimension of the Sandvik LH517, depicted from the side view. All dimensions are in millimetres (Sandvik, 2010).



Bucket	Size	A	B	C	D	E	F	G	H Right	H Left	J	J Right	J Left
Shark	7.0m3	6522	2896	3646	11055	11506	3066	3148	4798	4841	R7297	R3534	R3474

Figure 14. The dimension of the Sandvik LH517, depicted from the top view. All dimensions are in millimetres (Sandvik, 2010).

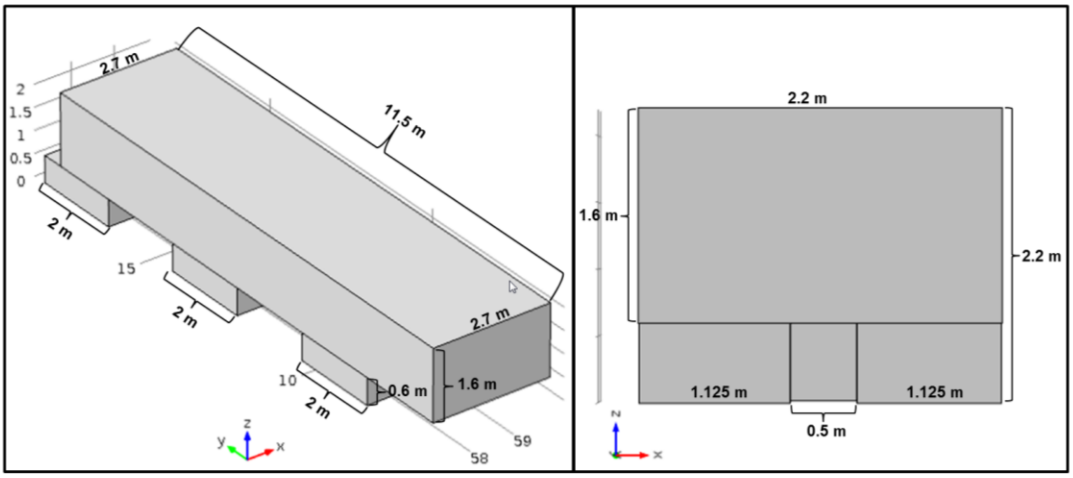


Figure 15. The dimension of the created LHD model in this research, in 3D view (left) and 2D view of the back part of the LHD (right).

For Scenario 3, those same three drifts were blocked with an air brattice instead of an LHD. Modelling this in COMSOL was simply done by cutting off the access to the workface of Drift 1, 2 and 4, so the airflow would only flow into Drift 3. Illustration of this model can be seen in Figure 16.

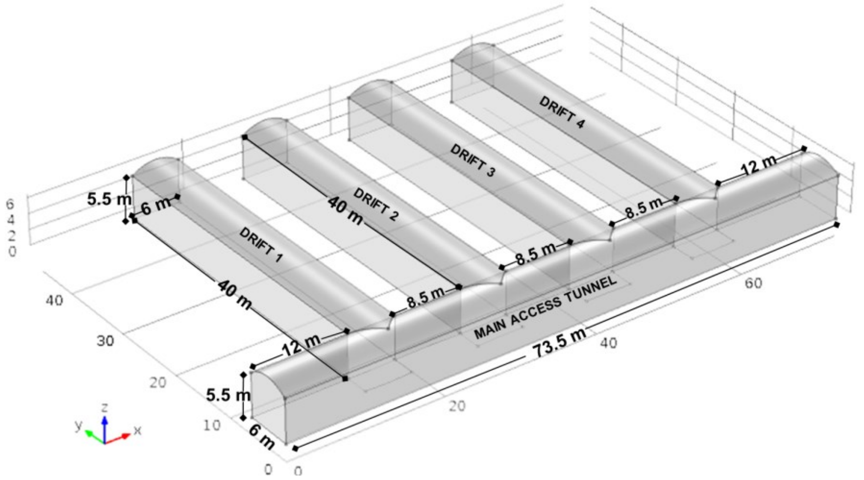


Figure 16. Geometry of the underground mine model in COMSOL for Scenario 3.

3.1.2 Setting the materials properties

After the geometry of the underground mine was already constructed, the next step that had to be done in the simulation is defining the materials of the properties. This is one of the basic requirements that have to be done in any simulation in COMSOL. By defining the material properties of the model, COMSOL program would be able to associate the model as a certain material and this would be the input during the calculation process.

In all of those three scenarios, the underground mine model acts as a component in the simulation. Since the main focus of the simulation is about air ventilation, therefore all of the material properties of these components were set as “air”. This was done by selecting the default material properties of “air” in the Materials Library in COMSOL program.

3.1.3 Setting the physics of the simulation model

The next step after defining the material properties is setting the physics of the simulation model. The main physics of the model was already set as “Turbulent Flow, k-ε” in the beginning of the modelling. What had to be further defined in this step is what domain of the component has the function as the inlet of air, the outlet of air, the walls and the fluid.

In the model, the inlet of the air is set as the inlet boundary. The same goes to the outlet boundary, where the air will exit the model through this outlet boundary. The default “wall functions” option in COMSOL was selected as the boundary condition of the wall in this model. Therefore, the surface roughness of the wall was assumed to be using the default settings from the COMSOL software. The wall functions will act as a solid wall and it models thin layers with high gradients in flow variables near the wall. illustration of those boundaries can be seen in Figure 17. There were only two boundaries that each acts as the inlet and the outlet. For the rest of the boundaries, they were defined as the wall of the model.

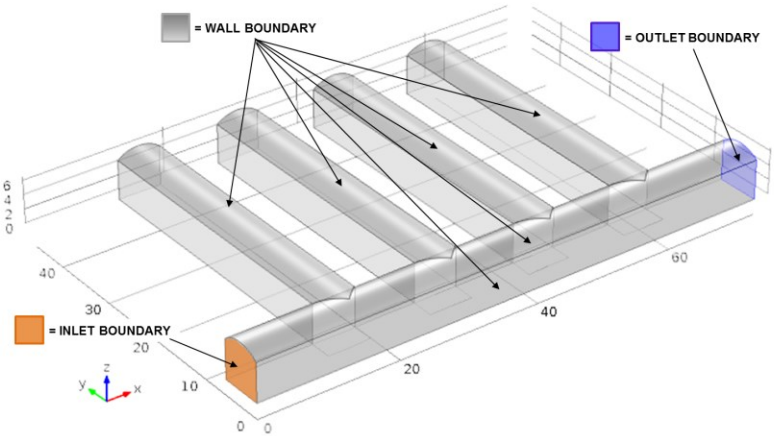


Figure 17. The definition of every boundaries of the simulation model in COMSOL.

There were several assumptions that were made for the inlet and the outlet boundary. In the inlet boundary, the initial speed was assumed to be 4 m/s. This number was considered reasonable enough. The reason for this refers to a notion in the German mining law, which states that the required minimum airflow speed is 0.5 m/s and the maximum airflow speed is 6 m/s (BBK I RWTH Aachen, 2012). While also according to that law, the maximum airflow speed in the longwall section of longwall mining is limited at 4.5 m/s. It was also mentioned before in Sub-chapter 2.2.5 where at the Pyhäsalmi mine, it is required to have a minimum air velocity of 1 m/s in order to dilute the dust particles (Rockplan, 2010). In the outlet boundary, the turbulent intensity was assumed as 5%, the turbulent length scale as 0.5 m, and the pressure boundary at the outlet as 120 Pa. These assumptions were made based on the numbers used by a study of CFD in a tunnel construction in Japan (Kanaoka et al., 2006).

3.1.4 Constructing the mesh

In this step, the model was discretized into smaller elements. These elements are the finite element mesh. The elements were simple geometrical shapes, in this case was the shape of a triangle. In the size properties in COMSOL program, the element size was set as “coarse”. This lets the COMSOL program to automatically define the size of each finite element to be not too small and not too large. There is a tendency for the size of the mesh near every edge and corner to be smaller because it is usually near this area where more accuracy and precision of the calculation result is needed.

In the boundary layer properties, the number of boundary layers was set as 5. The boundary stretching factor was set as 1.2. The thickness of the adjustment factor was set as 2.5. An example of the result in constructing the meshes in the model for Scenario 1 can be seen in Figure 18.

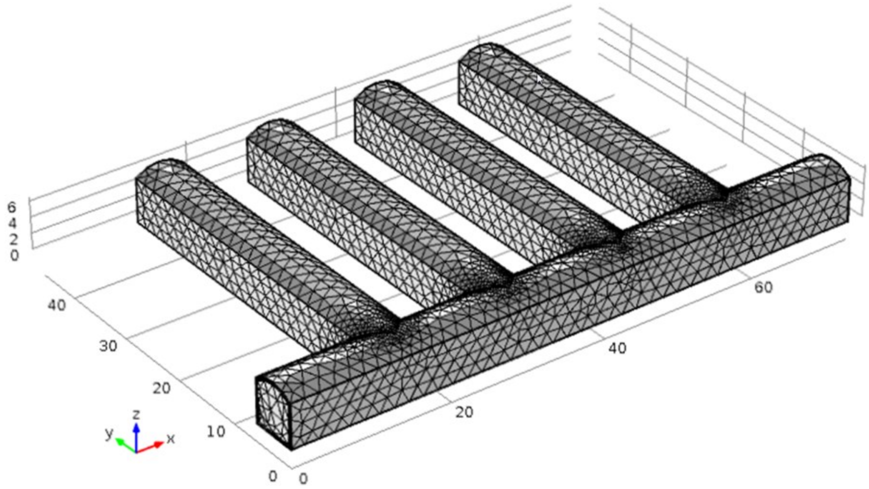


Figure 18. Model of Scenario 1 after the meshing process was done.

3.1.5 Running the study process

This step is the processing part of the simulation, which is the main core of the simulation. All of the previous steps before the study process are only for making the input for this processing step. During the simulation of all of those scenarios, the "Get Initial Value" function in COMSOL had to be conducted first in order to set the settings in the "Stationary Solver" window properties.

After that, the relative tolerance was set as 0.01. In the segregated node options, the chosen termination technique was "iterations". The number of iterations for each scenario was 600. This amount of iterations was considered sufficient enough because the convergence plot graph for each scenario showed that the error for each scenario was less than 0.01.

3.1.6 Plotting the results and calculating the operating cost

COMSOL program has the capability to plot the result of the simulation in various ways, such as by creating contour lines and plotting the magnitude of the result on the surface of the model. In this simulation, there were two types of variables obtained as the result of the simulation. Those two variables are the velocity magnitude (in m/s) and the pressure (Pa). The result of the velocity magnitude and the pressure were plotted into the model.

They were plotted with some visualisation and colouring effect to make the plotted results understandable for anyone to read it. The velocity magnitude throughout the model was plotted as arrow surface, surface contour, streamline and slice. Besides of the velocity magnitude, the pressure value was plotted as surface contour.

The cross section area of the tunnel was 37 m^2. Since the velocity at the inlet was 4 m/s, it means that the flow rate of the air from the inlet was 150 m^3/s. From here, the air power could be calculated. It was assumed that the system efficiency was 75%, the working day of the fan was 365 days, and operating hour per day was 24 hours. The electricity price that was used is 0.0915 €/kWh (Finpro ry, 2013). This was the electricity price for industrial consumers in Finland during the second semester of 2012.

By using these data, the total operating cost of the ventilation could be obtained. The price of the brattice cloth that was used to find the installation cost. The yellow ripstop cloth (brattice cloth) with a width of 4.3 meters costs $3 per meter (InfoMine USA, 2010). Labour cost was ignored during the calculation. All of the results related with ventilation modelling can be seen later in Chapter 4.

3.2 Research step for the scheduling optimization software

3.2.1 Debugging the scheduling optimization software

The scheduling optimization software that was used in this research was the first version of the program. The first version of the software was ready to be trialled on February 2014. During the trial, there were several data that was assumed and inputted into this software.

The assumed initial data that were used are:

a. There were 20 workfaces.

b. Five types of machines were used (scaler, shotcreter, bolter, driller, and charger).

c. Two dump sites were available.

After that, the software executed the processing of the data by using the software's algorithm. The scheduling optimization software then produced several Gantt charts as explained before in Sub-chapter 2.3.1. It was important during the trial that all of the input data have to be placed correctly; otherwise the program could not process the input data. The results of this debugging phase can be seen in Sub-chapter 4.2.1 about "*error issues*".

3.2.2 Reconciling the data from the weekly plan of Kittilä mine

The data received from Kittilä was inputted into Microsoft Excel 2010 and were grouped into different worksheets. This was for making the inputting process into the scheduling optimization software easier. The weekly plan of Kittilä during 4-10 September 2013 was used as the input data for the software (the picture of this weekly plan can be seen in Figure 9 Sub-chapter 2.3.4). The details of these input data can be seen in Appendix 1. There were 35 workfaces, 3 machines set, 7 dumping locations, and 7 types of machines. The workload at each workface and the machine operating data were based on the data from that Kittilä's weekly plan. All of the equipment were assumed to be initially located at Workface #15 because it is the closest workface to the level where the main underground workshop is located. The location of those 35 workfaces in the layout of Kittilä mine can be seen in Figure 19.

After all of the data had been reconciled, the data was exported manually into text file format. This is because the software could only receive input data in this type of format. After that, the program was executed to obtain the Gantt charts. All of the Gantt charts were based on scheduling by sharing machines and workface sorting.

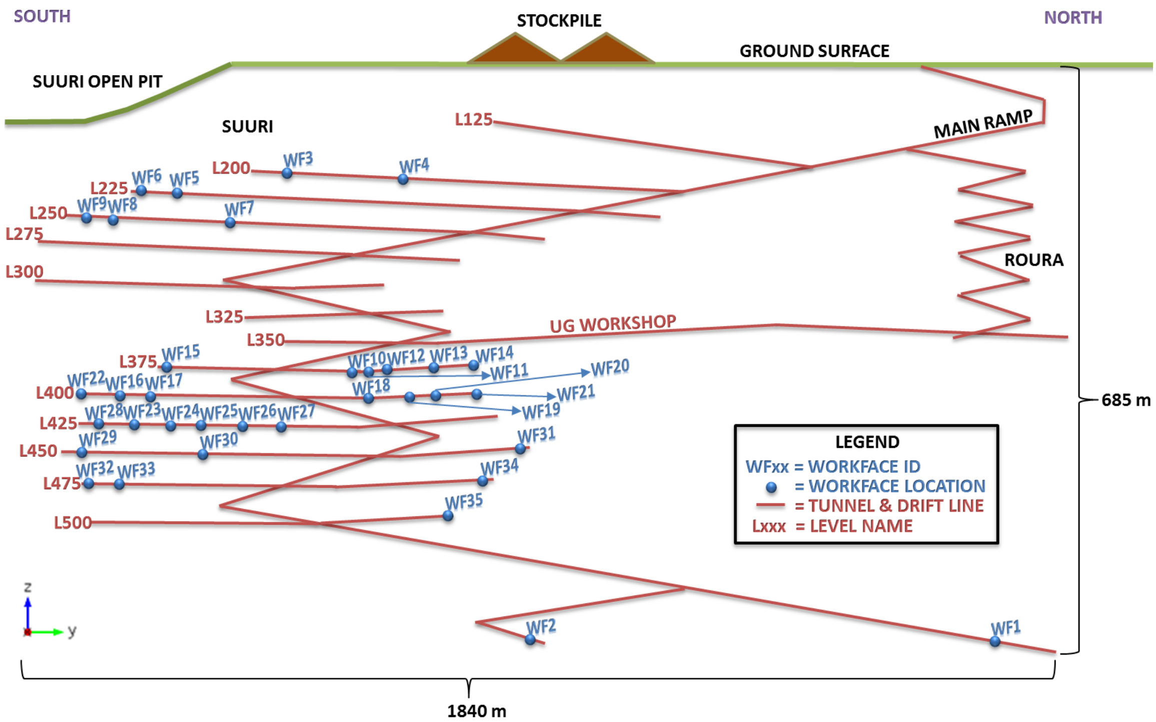


Figure 19. The layout of Kittilä mine with the locations of the workface (indicated by WF which stands for workface; L is the name of the level) that were used as the input data.

3.2.3 Creating the scenario

After being able to produce Gantt chart schedules for the weekly plan, there were 11 scenarios created as a modification from the studied weekly plan of Kittilä in the previous stage. The list of the scenarios can be seen in the Table 5, where each scenario has a parameter that was modified. For example, in the case of Scenario 1 in Table 5, it was assumed that the access to Level 250 was suddenly cut off due to an unexpectedly roof collapse at this access. This event in Scenario 1 means modifying the original data from Kittilä's weekly plan. The modification that was done is by deleting several workfaces with ID number 7, 8, and 9 from the original workface data set. The reason for deleting these is because these workface were located at Level 250.

Another example, in Scenario 2, it was assumed that in the second equipment fleet (or in other words, "machine set 2"), there were one bolter and one charger equipment that were having a breakdown. This breakdown event made those equipment not available, so those equipment were simply stated as "not available" in the input data (by editing the "Machine_Set.txt"). These assumptions were made as realistic as possible and it applies for the whole 11 scenarios for the scheduling optimization software.

Table 5. The new scenarios created for trial in the scheduling optimization software.

Scenario Number	The assumed scenarios
1	Access to Level 250 was not available, thus eliminating workface #7-#9.
2	One bolter and one charger were not available in machine set 2.
3	Dump site B was not available because it was already full.
4	Only two machine sets were available.
5	No drilling workload at Workface 10, 15, 20 and 25.
6	All machine initial location were partially at workface 3 and 10.
7	Only 2 dump sites were available; with only 1 scaler, 1 shotcreter and 1 charger were available (so only all machines in machine set 1 that were normally available, but the availability of the rest of the equipment in other fleets were adjusted)
8	No scaling workload at workface 12, 25, 30; no shotcrete workload at workface 18, 27, 34; no drilling workload at workface 3, 20, 33; no charger workload at workface 5, 9, 14; only one shotcreter was available, dump site D and E were not available
9	Combination of scenario 1, 2 and 5
10	Combination of scenario 1, 4 and 6
11	Workface 3, 5, 8, 9, 12, 13, 16, 17 , 19, 20, 23, 25, 28, 30, 31, 33 and 34 were deleted, only 2 dumps were available (Dump A and waste dump)

3.2.4 After running the scheduling optimization software

The Gantt charts produced from each scenario were then compiled and compared to each other. These results and the process to obtain the Gantt chart would be evaluated, in order to evaluate the functionality of this scheduling optimization software. The result from the “LHD and truck” scheduling optimization were in the form of text file. This format file was not very convenient to read, as can be seen in Figure 7 Sub-chapter 2.3.3 page 23. Therefore, the results were copied from text file and formatted into an Excel file in Microsoft Excel to make it looked more convenient to read. All of the results and discussion could be seen in Sub-chapter 4.2.

4 Results

4.1 Mine ventilation modelling

During the beginning of modelling the ventilation in COMSOL, there were several difficulties that were encountered. For example, there were problems with creating the desired geometry model in COMSOL. The LHD model for Scenario 2 was first created in AutoCAD 2010 program to get a detailed design of the LHD (with precise dimension, edges and faces of the LHD adapted from the Sandvik LH517). When this LHD model was imported into COMSOL and discretised into finite meshes, there were too many small meshes, which caused long calculation time and errors during the meshing phase. The popped up warning message in COMSOL program said "*failed to respect boundary element on geometry edge*" and "*edge is much shorter than the specified minimum element size*".

Thus, to avoid that situation, the LHD model was changed into a simpler design with fewer edges. This successfully avoided the meshing problem that was experienced before. The illustration of this case can be seen in Figure 20.

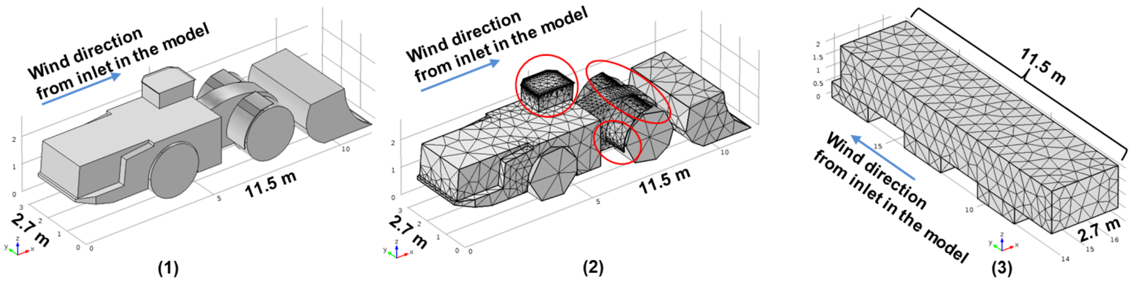


Figure 20. (1) The initial LHD design in COMSOL; (2) The mesh result of the initial design, the red circles show a lot of small meshes; (3) The simplified version of the LHD model that was successful and used in this research.

The following sub-chapters show the results that were obtained from the simulation in COMSOL program.

4.1.1 Arrow surface of the velocity magnitude

The arrow surface plot depicts the direction of the air velocity that flows under the roof surface. In the result of Scenario 1 (all drifts were open), arrows throughout the roof surface of the model show that most of the air velocity is flowing at the main access tunnel. It can be seen in the main access tunnel where the big arrows show the wind flowing from the inlet boundary to the outlet boundary. Some of the air also flows into Drift 1, 2, 3 and 4. From all of these drifts, most of the wind easily flowed into Drift 1 because it is the closest drift to the inlet. The air flow pattern in Drift 1 is quite consistent in one direction. After Drift 1, the airflow in Drift 2, 3, and 4 have less velocity and more scattered arrow patterns. This can be seen in Figure 21.

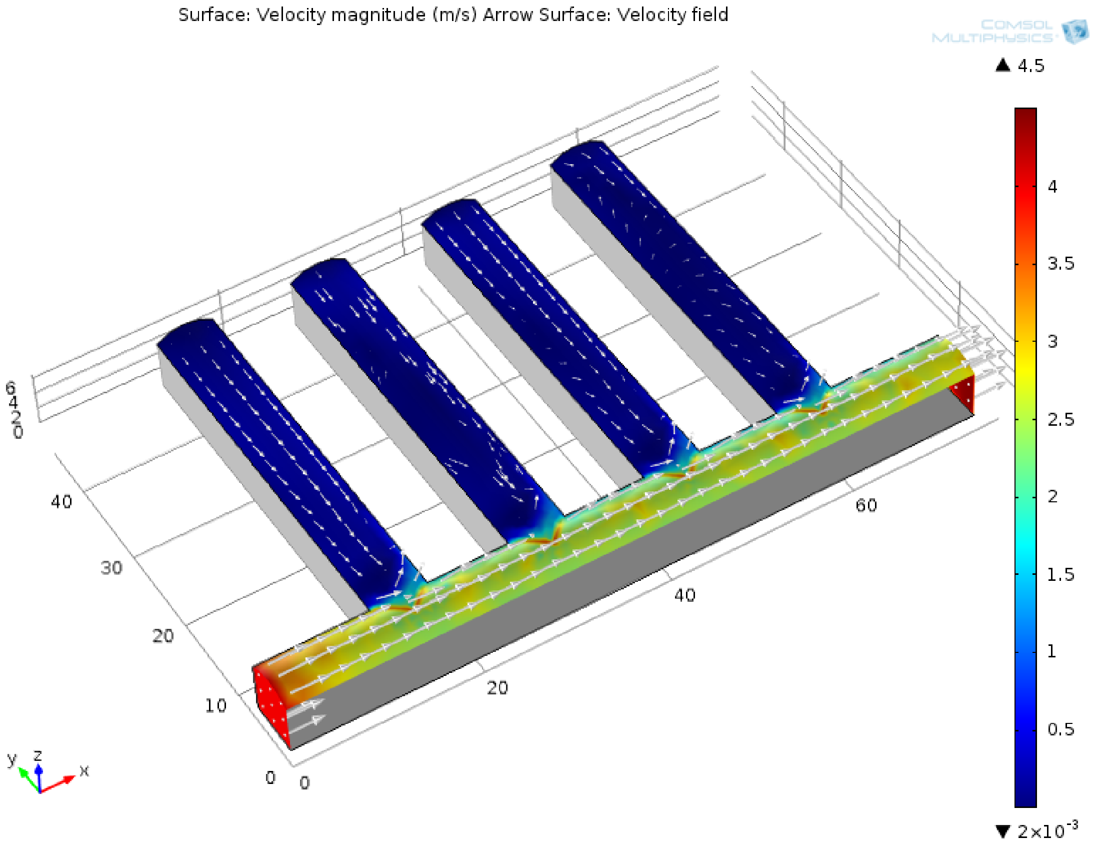


Figure 21. The velocity magnitude (m/s) and the arrow surface of velocity field at the roof surface of Scenario 1 (all drifts were open, without any blocking).

In the case of Scenario 2 (all drifts were blocked with LHD except at Drift 3), the arrow surface plot seems similar to the case of Scenario 1. But, the velocity magnitude in the main access tunnel is higher than the one in Scenario 1. Besides of that, in Scenario 2 Drift 1, there are less arrows and the velocity magnitude is less. It could also be seen in Scenario 2 Drift 3 where there are more arrows compared to Scenario 1. This means that the velocity field in Scenario 2 Drift 3 is higher than in Scenario 1. The instalment of LHD in Drift 1 and Drift 2 seems to increase more turbulence in the intersection between those two drifts and the main access tunnel. This then makes more air to just keep flowing in the main access tunnel rather than entering the drifts. However, based on the velocity magnitude (this can be seen by the dark blue colour), there is a small difference between Scenario 1 and Scenario 2 in all of the drifts. The illustration of this case can be seen in Figure 22.

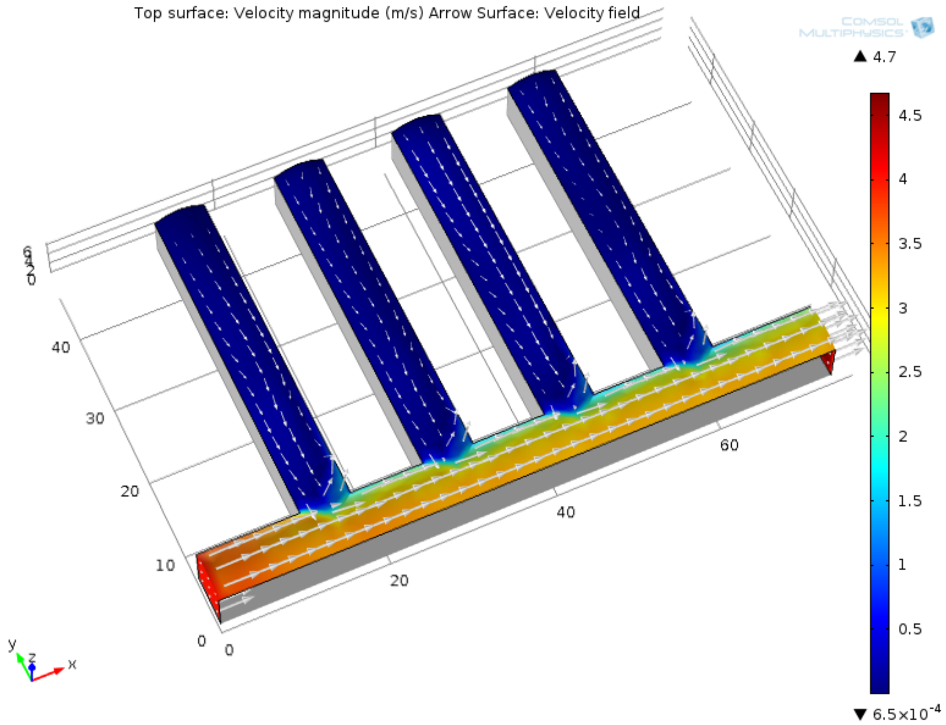


Figure 22. The velocity magnitude (m/s) and the arrow surface of velocity field at the roof surface of Scenario 2 (all drifts were blocked with LHD except at Drift 3).

For Scenario 3 (all drifts were blocked with air brattice, except at Drift 3), it is clearly noticeable that there is more air flowing into Drift 3. This is due to the practice of totally blocking the airflow into Drift 1, 2, and 4. Drift 1, 2 and 4 show no arrows at all because there is no velocity field in this drifts. This can be seen in Figure 23.

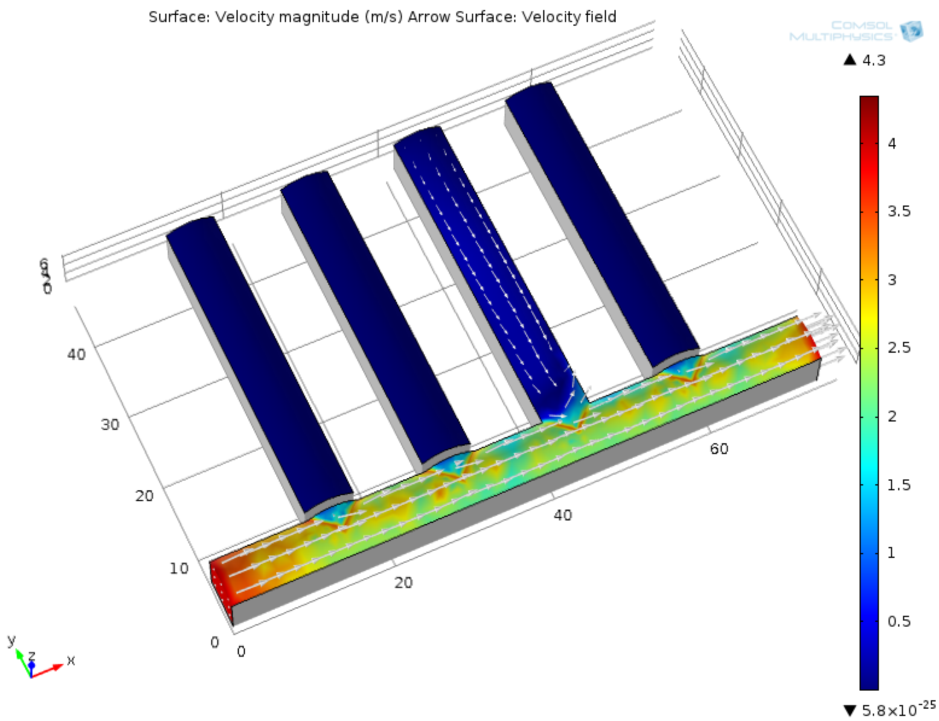


Figure 23. The velocity magnitude (m/s) and the arrow surface of velocity field at the roof surface of Scenario 3 (all drifts were blocked with air brattice, except at Drift 3).

In general, the practice of blocking Drift 1, 2 and 4 with air brattice is more effective to maximize the airflow into Drift 3. This happened in Scenario 3. It can be seen in Scenario 1 where air freely moves into Drift 2 which did not need any airflow at all. The practice of blocking the drifts with LHDs gave some difference, but not too significant compared to the performance of air brattice because the LHDs did not totally block the whole cross section of the drift.

4.1.2 Velocity magnitude contour

In this plot, the value of the velocity magnitude could be easily seen throughout the surface of the model because contour lines could display the value of this variable (the velocity magnitude). COMSOL program has an option in this plot to label certain contour lines automatically. These labels make the identification of the value of velocity magnitude easier in the model.

It can been seen for the model of Scenario 1 in Figure 24, that the velocity magnitude at all of the drifts ranges from around 0.227 m/s until 0.676 m/s. In the main access tunnel, the velocity magnitude near the inlet is 3.67 m/s, and as the airflow pass through the main access tunnel, the velocity magnitude decreases to 3.2 m/s near the intersection with Drift 1, and then becomes 2.02 m/s after the intersection with Drift 4.

It has to be noted that there is some high velocity magnitude at the edge of this V-shaped pocket (look at those orange-coloured V located at the front of every intersection in Figure 24). The high velocity magnitude in this V-shaped pocket happened because of the accumulation of airflow from two directions:

- The first direction is airflow from the inlet going to the outlet, flowing in the main access tunnel.
- The second direction is return airflow from the drift, which went out from the drift and went into the main access tunnel.

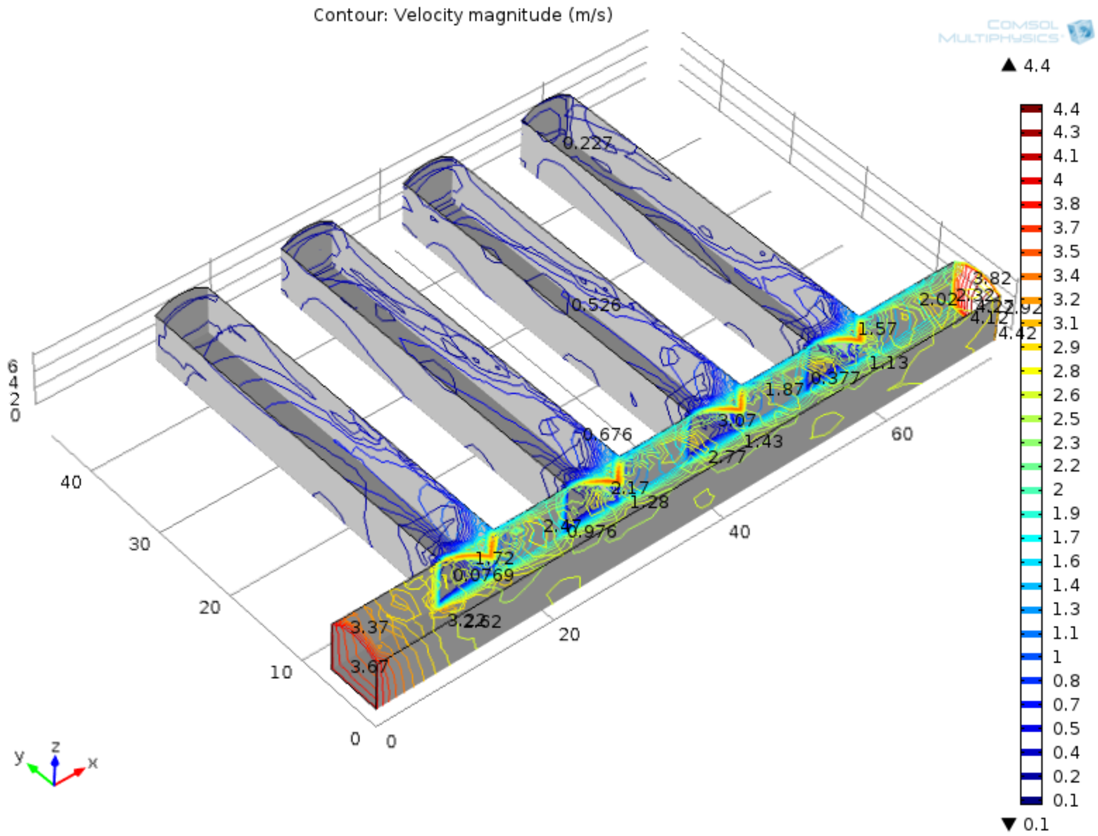


Figure 24. The contour of the velocity magnitude (m/s) throughout the model of Scenario 1 (all drifts were open, without any blocking).

Figure 25 shows the velocity magnitude of Scenario 2. All of the LHDs are depicted as a magenta-coloured blocks in the entrance of Drift 1, 2 and 4. It is interesting to see how actually there is some influence by parking the LHD at the entrance of the drift. Compared to Scenario 1 Figure 24, there are not a lot of contour lines in Drift 1, 2, 3, and 4. The contour lines in the main access tunnel also show that the velocity magnitude value in the main access tunnel is higher than those in Scenario 1. However, the velocity magnitude in all of the drifts is lower than 1 m/s.

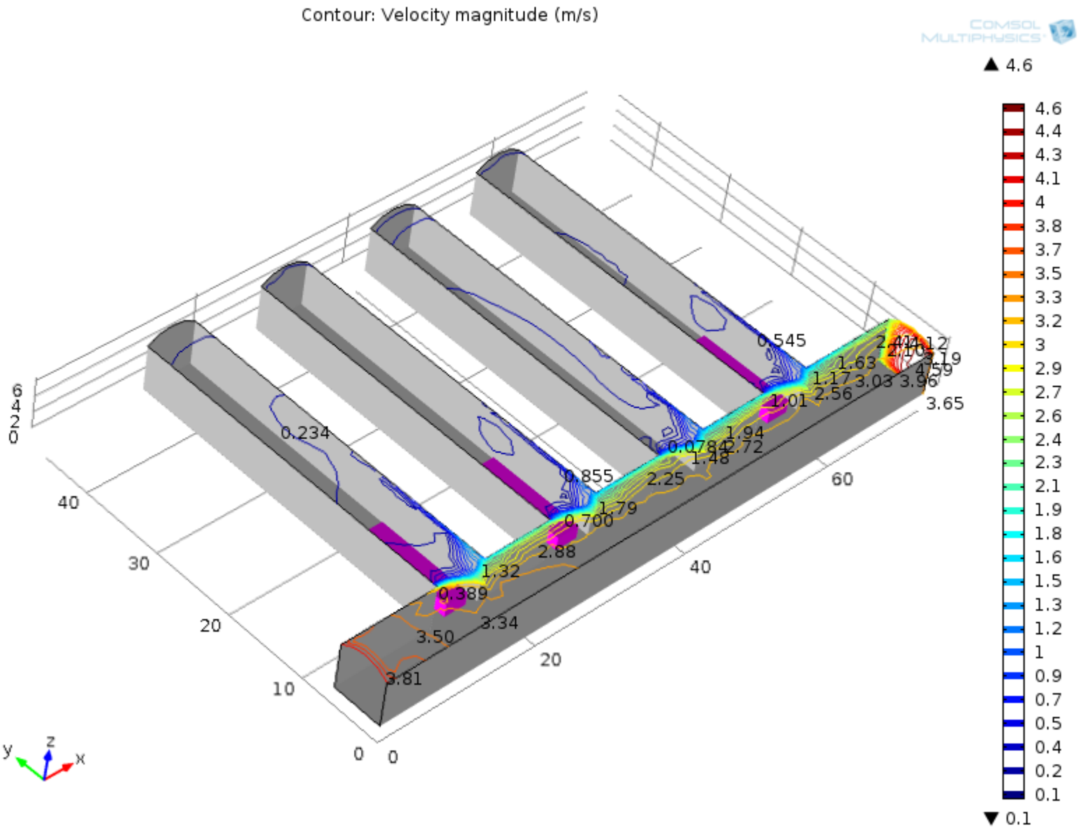


Figure 25. The contour of the velocity magnitude (m/s) throughout the model of Scenario 2 (all drifts were blocked with LHD except at Drift 3).

At the outlet boundary in Figure 24 and Figure 25, COMSOL program made some labels which showed the velocity magnitude of above 4.12 m/s in partial areas of the outlet surface. The average airflow velocity at the outlet surface in both Scenario 1 and Scenario 2 are still 4 m/s despite of these partial areas with higher velocity (the same value as the inlet airflow velocity).

The outlet boundary was relocated further away for 20 meters from the original location in the model to confirm the higher velocity at the outlet surface. The relocation was only done for the model of Scenario 2. After this modification was done, it showed that at that location of the original outlet, the velocity magnitude is only around 2.3 m/s. The illustration of this modification can be seen in Figure 26. Most of the partial areas with high airflow velocity are only at the middle part of the outlet surface. This means that the cause of this high velocity is only because that area has less friction and resistance for airflow to flow.

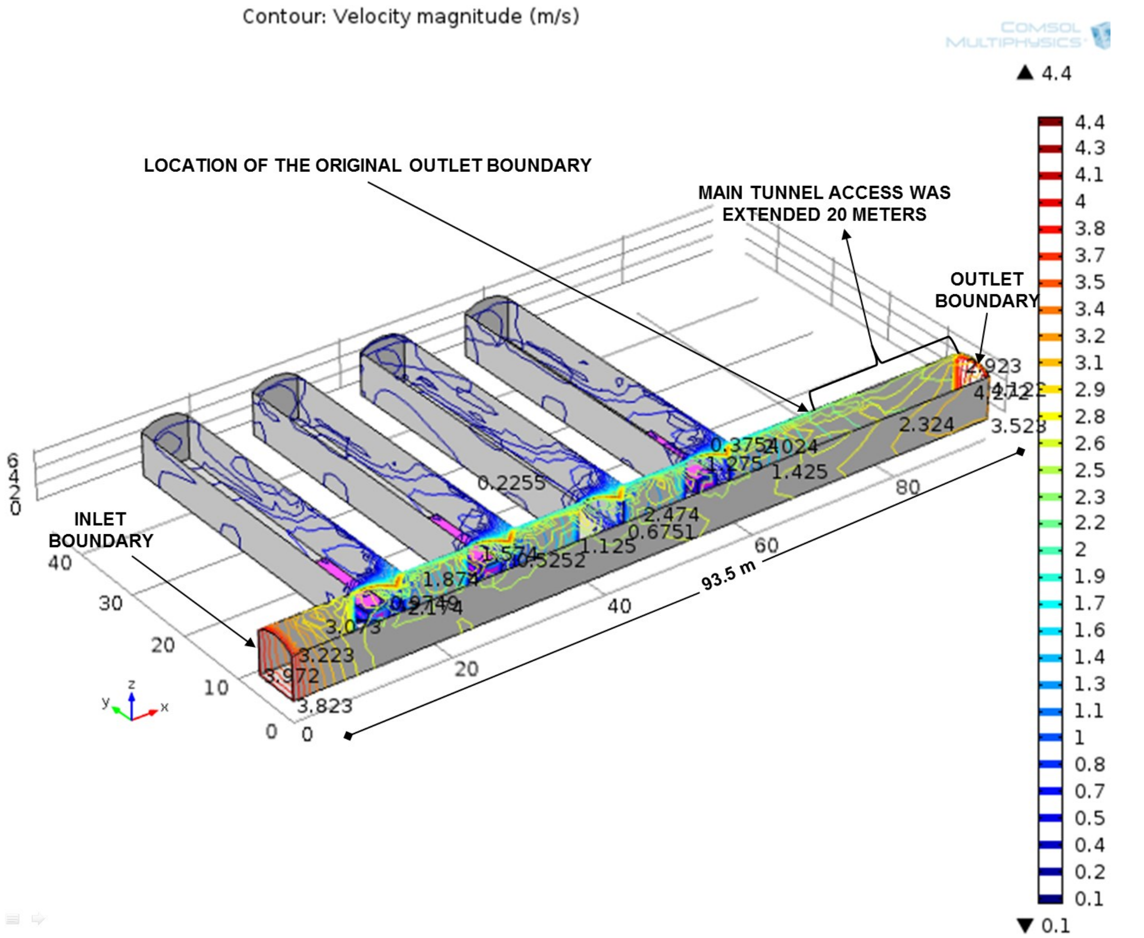


Figure 26. The model of Scenario 2 with the outlet boundary located 20 meters further away from the original position (all drifts were blocked with LHD except at Drift 3).

The contour plot of the velocity magnitude in Scenario 3 can be seen in Figure 27. Since no airflow could enter Drift 1, 2 and 4, there is no contour plot in these drifts. The contour lines in the main access tunnel has orange colour, which means that the velocity magnitude in the main access tunnel is generally around 3 m/s. This is much higher than the velocity magnitude of the main access tunnel in Scenario 1 and 2. The only drift that has contour lines is in Drift 3, which is not blocked at all. But, the velocity magnitude is very low. Even the velocity magnitude near the end of the drift is around 0.072 m/s. It shows that the majority of the airflow just keeps moving in the main access tunnel. In Scenario 3, it can be seen that only at the front of Drift 3 has occurrence of turbulence. This is indicated by more blue contour lines at front of Drift 3 compared to other drifts.

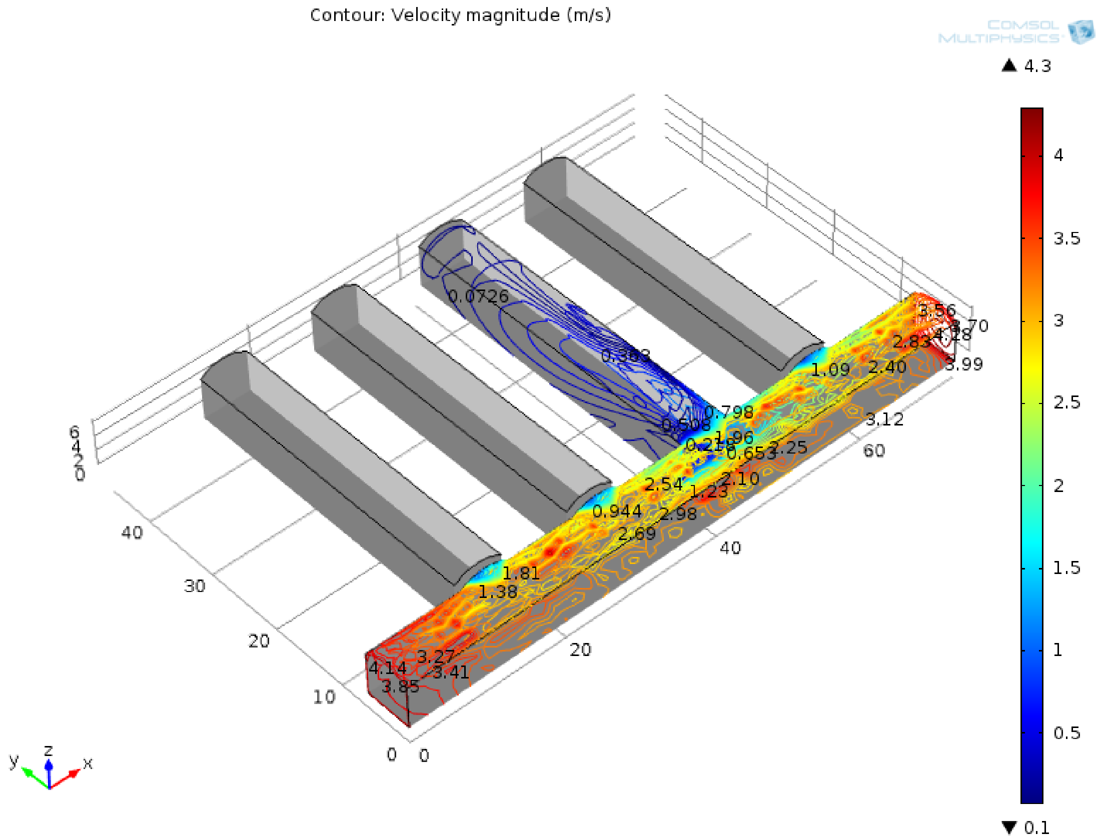


Figure 27. The contour of the velocity magnitude (m/s) throughout the model of Scenario 3 (all drifts were blocked with air brattice, except at Drift 3).

4.1.3 Streamline

The streamline plot for all scenarios were coloured by their velocity magnitude and the width of every streamline were proportional to the turbulent viscosity. The streamline plot of Scenario 1 (Figure 28) shows that there is a lot of turbulence at near the end of Drift 1 and at the tunnel segment between the inlet and Drift 1. The further away the location of the drift, the lesser is the occurrence of turbulence. This is noticeable in Drift 4 where there is less turbulence than in Drift 1, 2 and 3.

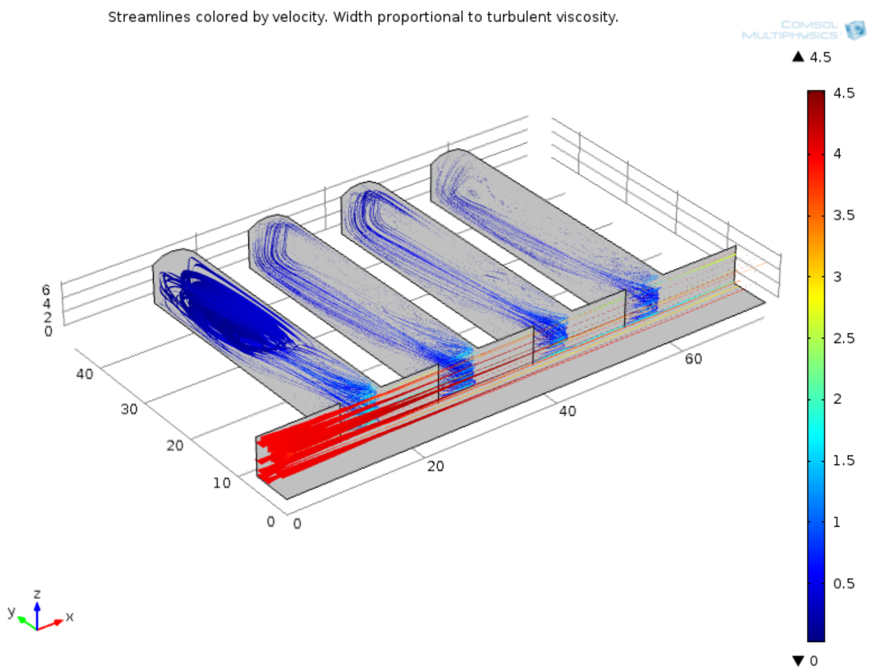


Figure 28. Streamline plot of the velocity field (m/s) in the model of Scenario 1 (all drifts were open, without any blocking).

Figure 29 shows the streamline plot in Scenario 2. It can be easily seen that compared to Scenario 1, most of the airflow in the drifts experienced more turbulence. This is shown by the thicker streamlines in Drift 1, 2, 3 and 4 compared to Scenario 1. This is due to the positioning of LHD at the front of the drift. The airflow had to encounter more friction against the surface of the LHD, thus having more turbulence instead of flowing in a less turbulence state.

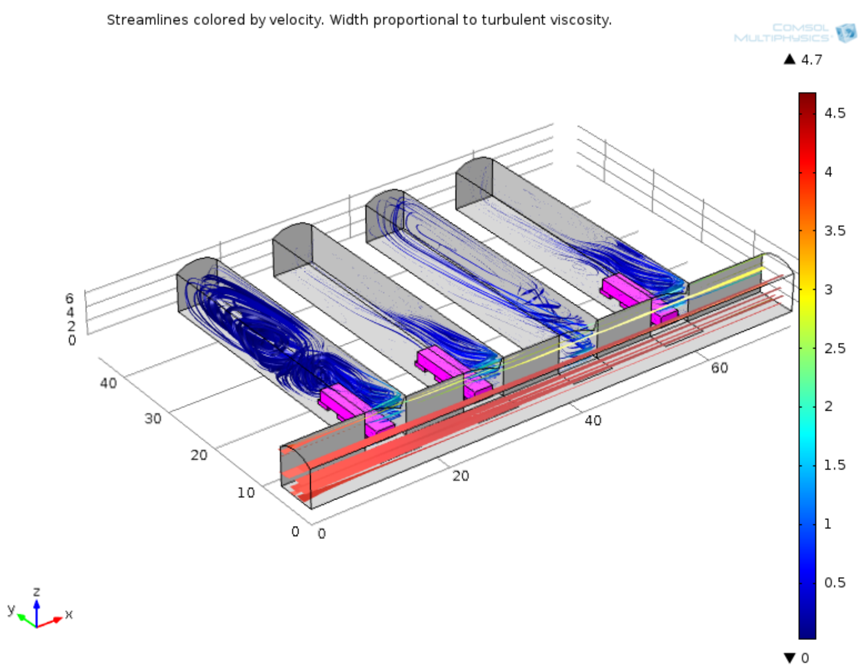


Figure 29. Streamline plot of the velocity field (m/s) in the model of Scenario 2 (all drifts were blocked with LHD except at Drift 3).

Figure 30 shows the streamline plot for Scenario 3. There is no turbulence at Drift 1, 2, and 4 because the drift's entrances are blocked with air brattice. In this way, Drift 3 has more turbulence than those in Scenario 1 and Scenario 2 because it has more airflow coming in.

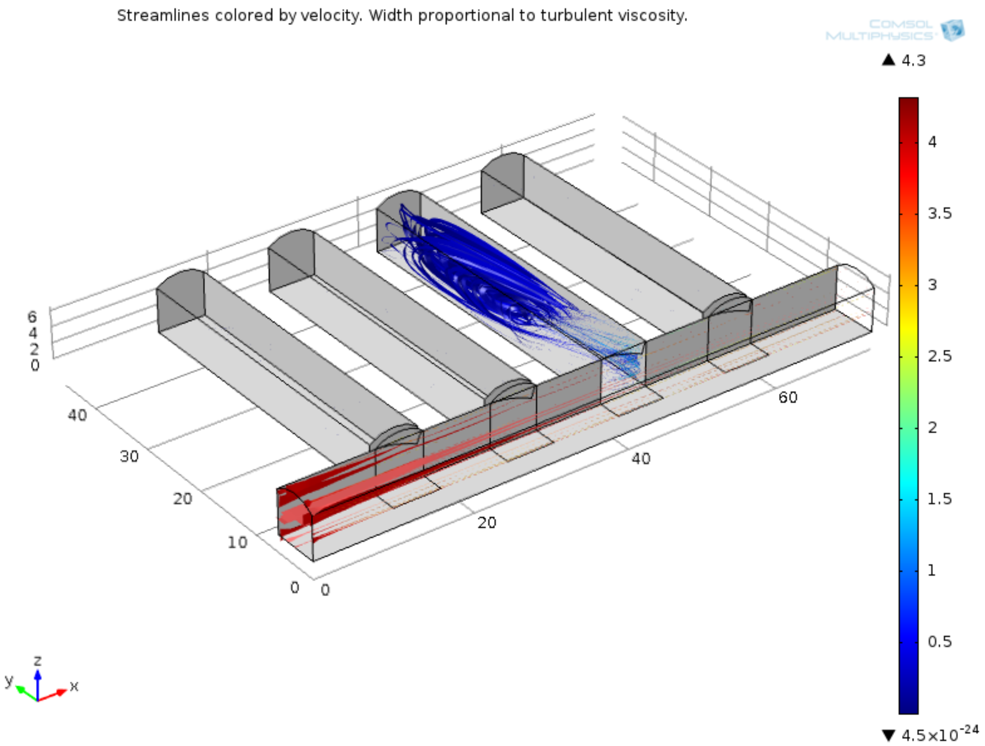


Figure 30. Streamline plot of the velocity field (m/s) in the model of Scenario 3 (all drifts were blocked with air brattice, except at Drift 3).

4.1.4 Vertical slice of the velocity magnitude

Slice velocity magnitude plot is a nice way to find out the value of velocity magnitude in certain locations in the 3D space of the model. It is because it divides the model into several slices. These slices have a fixed distance from one slice to another slice.

Figure 31 shows the slice plot of velocity magnitude in Scenario 1. It can be seen from all of those slices, that on the right hand side of the airflow flowing from the inlet, most of the velocity magnitude is around the range of 4 m/s. All of the slices that go through the centre axis of each drift are dominated with the blue colour. This colour means the velocity magnitude in every drift is around 0.5 m/s. At every intersection between the drifts and the main access tunnel, the slices are mostly dominated with the red colour, but the left hand side of the edge has a gradation colour from red to blue. It seems that although the entrance of every drift is open, there is not a lot of airflow entering the drifts. The airflow just mainly stays in the main access tunnel and only a few enters the drifts.

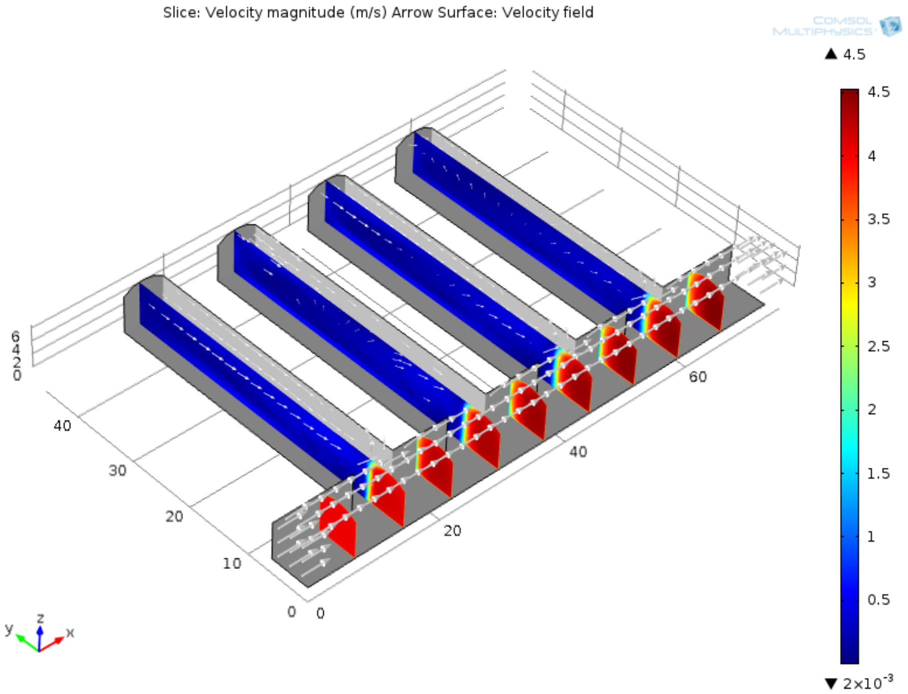


Figure 31. Vertical slice plot of the velocity magnitude (m/s) in the model of Scenario 1 (all drifts were open, without any blocking).

Figure 32 shows the slice plot of velocity magnitude in Scenario 2. The plot seems similar to the plot in Scenario 1. The only exception in Scenario 2 is that the blue region in the slice that goes through Drift 3 has a slightly brighter blue colour compared to the slices in Drift 1, 2, and 4. This means that all of the drifts that have an LHD parked at the drift entrance have lower velocity magnitude than the drift without an LHD.

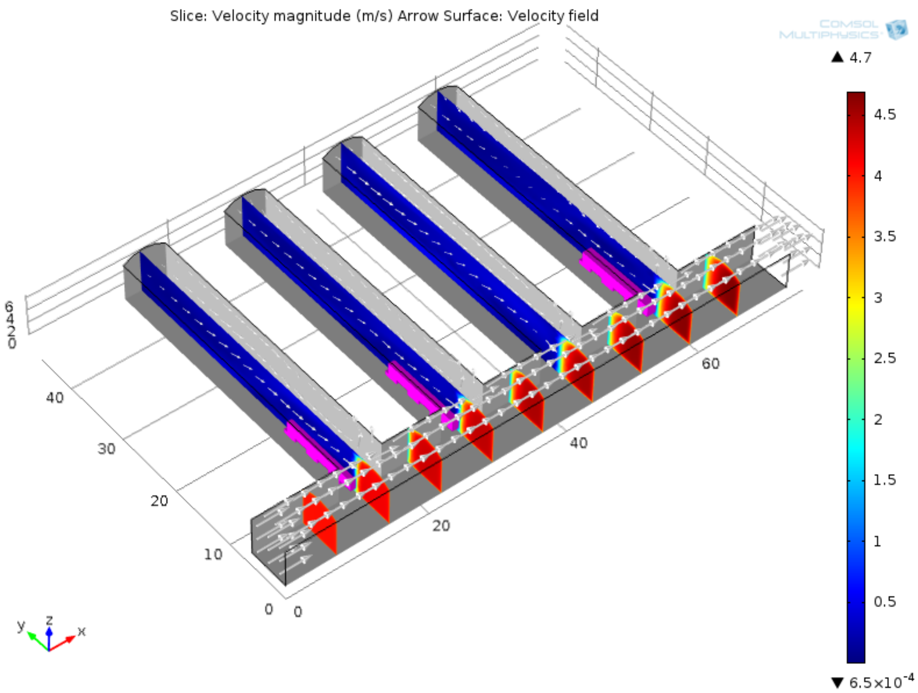


Figure 32. Vertical slice plot of the velocity magnitude (m/s) in the model of Scenario 2 (all drifts were blocked with LHD except at Drift 3).

Figure 33 shows the slice plot of velocity magnitude in Scenario 3. The slice plot in Scenario 3 looks very similar to the slice plot of Scenario 2. The only difference is that since there is no airflow at all in Drift 1, 2, and 4, therefore the slice plot in these drifts are darker than the slice plot in Scenario 2.

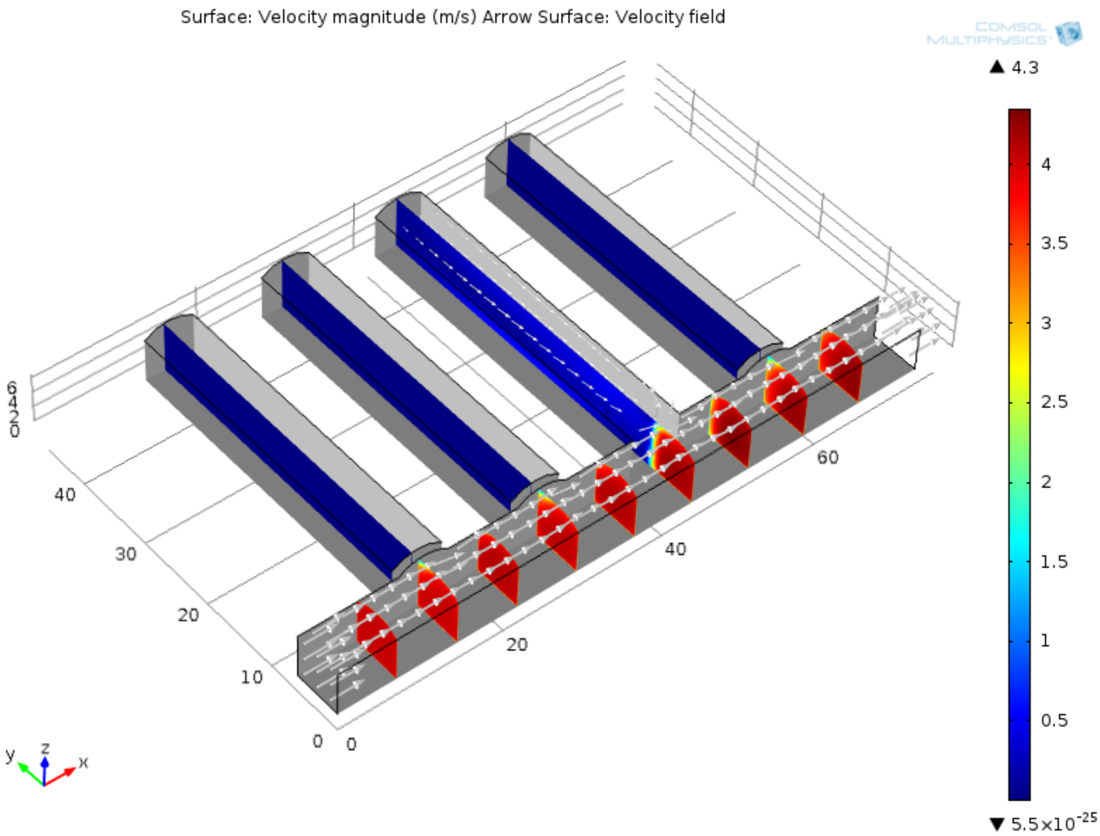


Figure 33. Vertical slice plot of the velocity magnitude (m/s) in the model of Scenario 3 (all drifts were blocked with air brattice, except at Drift 3).

4.1.5 Horizontal slice with arrow volume

In order to find the velocity magnitude throughout one particular horizontal layer of the model, the horizontal slice with arrow volume plot was created. All of the horizontal slice plots were located at 3 meters above the ground. Figure 34 shows the horizontal slice of Scenario 1. In Scenario 1, most of the airflow went through the main access tunnel. The airflow magnitude into the drifts is only around 0.5 m/s with some near the intersection reached around 1 m/s.

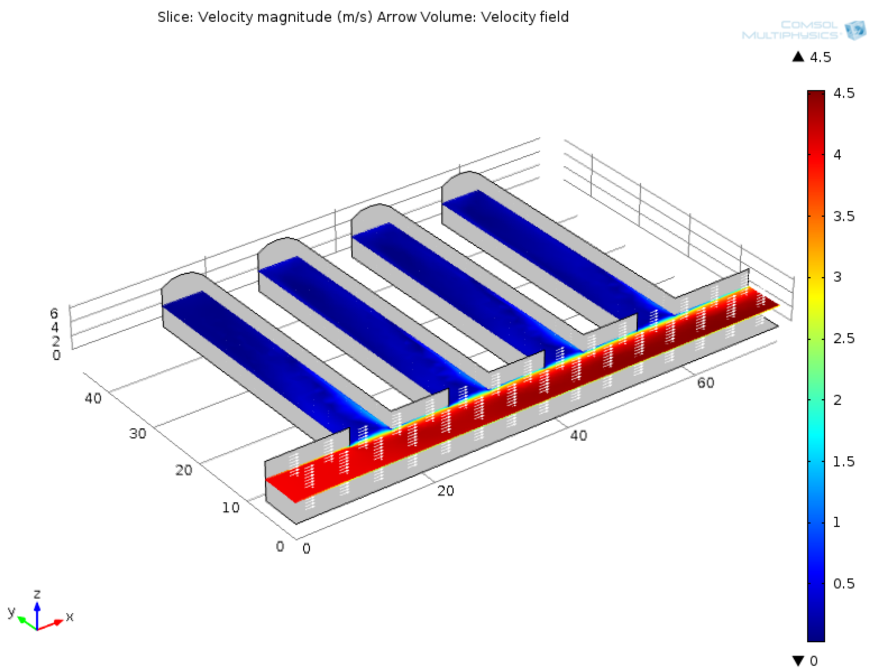


Figure 34. Horizontal slice plot of the velocity magnitude in the model of Scenario 1 (all drifts were open, without any blocking).

Figure 35 shows the horizontal slice plot of the velocity magnitude of Scenario 2. In this scenario, the horizontal slice plot also shows similar results as the horizontal plot in Scenario 1. In the scenario plot of Scenario 3, it also has the same result as in Scenario 1 and Scenario 2. This can be seen in Figure 36. In comparison to Scenario 2, Scenario 3 does not have airflow at all in Drift 1, 2 and 4 because it is totally blocked with air brattice.

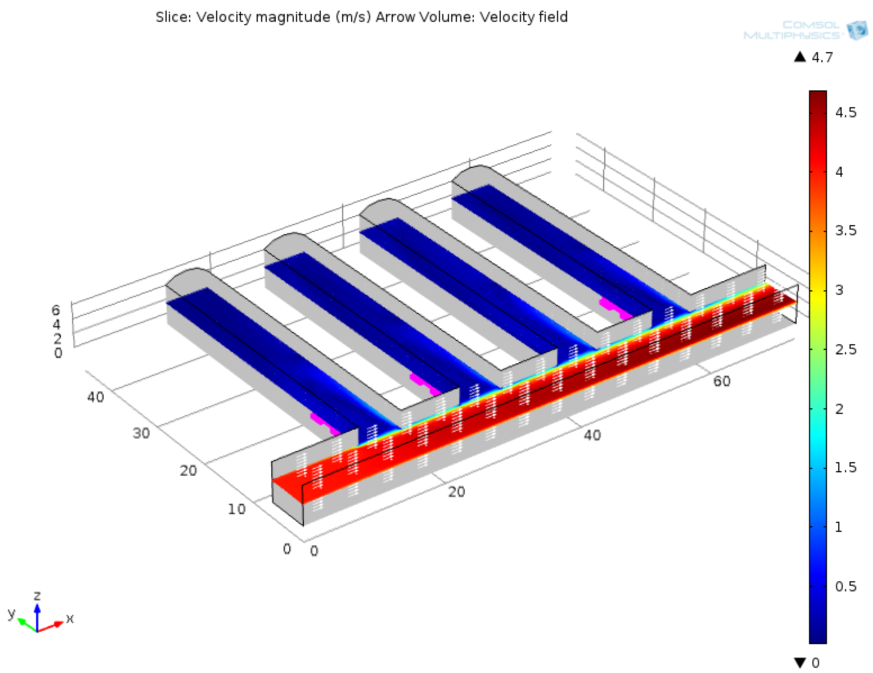


Figure 35. Horizontal slice plot of the velocity magnitude in the model of Scenario 2 (all drifts were blocked with LHD except at Drift 3).

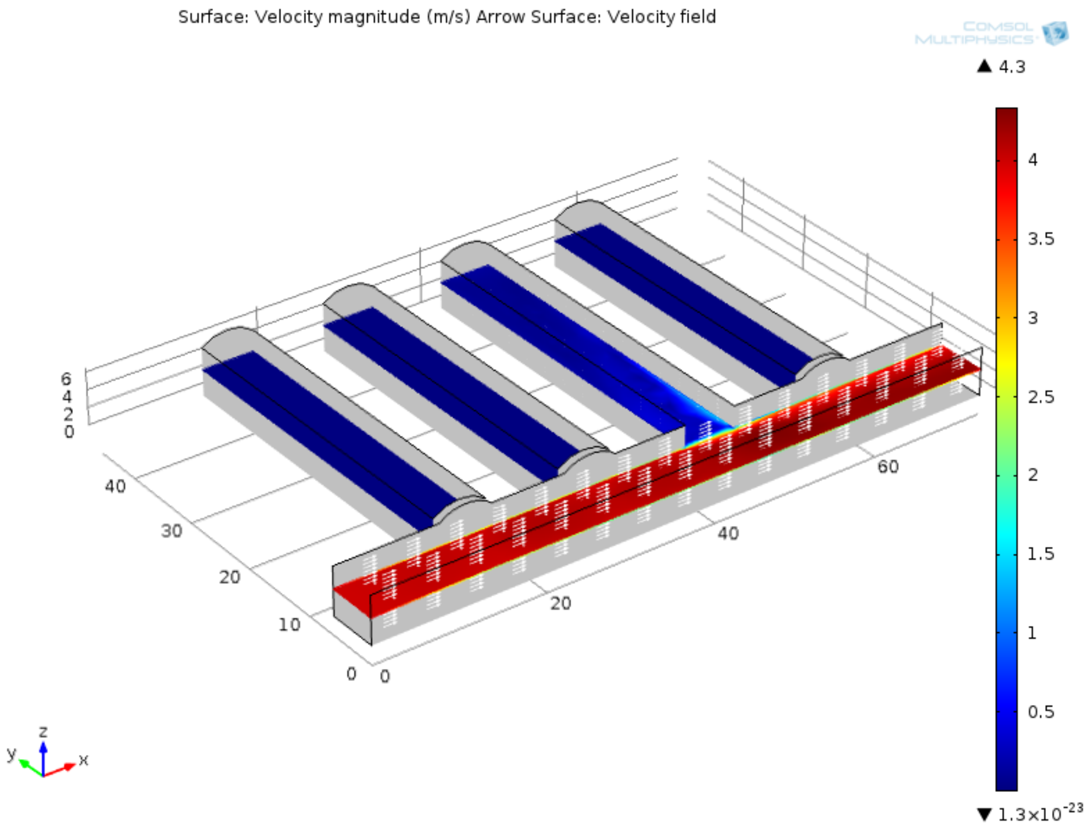


Figure 36. Horizontal slice plot of the velocity magnitude in the model of Scenario 3 (all drifts were blocked with air brattice, except at Drift 3).

4.1.6 Pressure contour

The plotting concept of pressure contour plot is just the same as the velocity magnitude contour plot. In this sub-chapter, the values of pressure as the result of the simulation throughout the model were plotted in the form of contour lines.

Figure 37 shows the pressure contour plot in the model of Scenario 1. Since the air pressure at the outlet boundary was set as 120 Pa in the settings before running the simulation, it could be easily seen that there is a decline of pressure from the inlet boundary to the outlet boundary. According to the contour lines, the pressure near the outlet boundary is around 120 Pa. While near the inlet boundary, the pressure is at around 125 Pa.

During the plotting of surface contour, all of the level labels in the plot (indicated by the black labels on the model in Figure 38) were automatically generated by COMSOL program. They could not be set manually. The legend scale on the bottom side of Figure 37 depicts the value of each contour lines based on their colours, ranging around 120 Pa to 124.9 Pa.

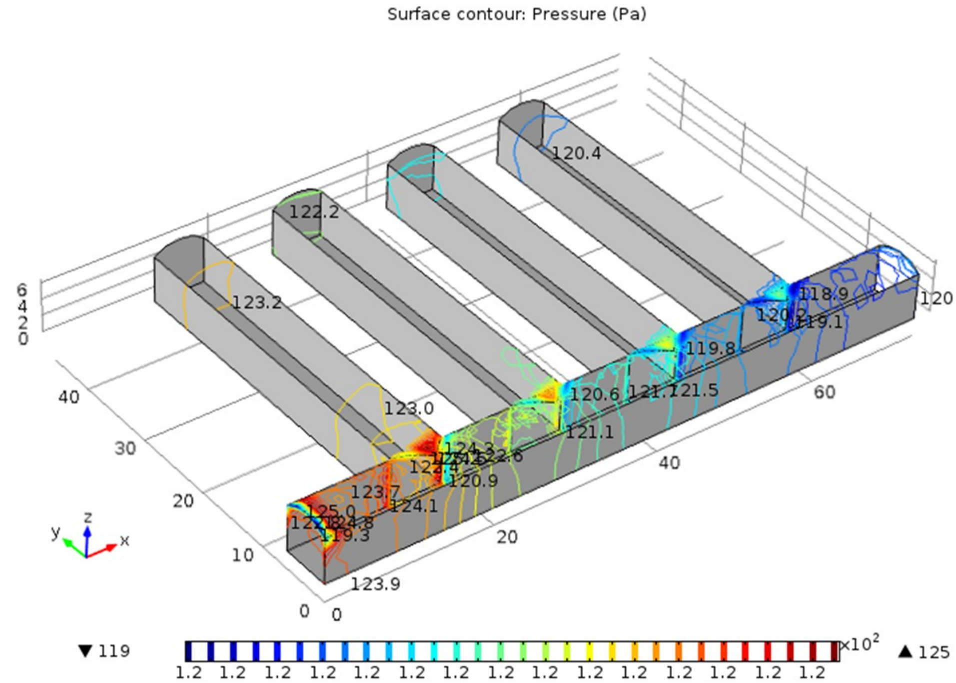


Figure 37. Surface contour of the pressure quantity (Pa) in the model of Scenario 1 (all drifts were open, without any blocking).

In Scenario 2 (Figure 38), the plot of the pressure contour lines is also similar like in Scenario 1. It has a decline of pressure from the inlet boundary to the outlet boundary. The value of the pressure throughout the model is also similar to Scenario 1.

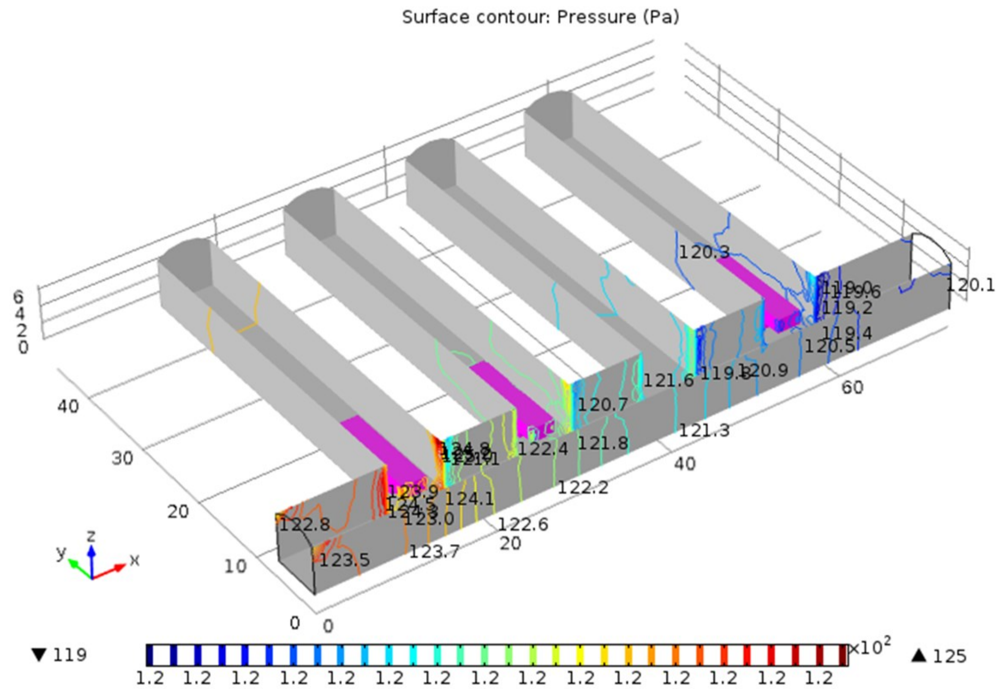


Figure 38. Surface contour of the pressure quantity (Pa) in the model of Scenario 2 (all drifts were blocked with LHD except at Drift 3).

Figure 39 illustrates the pressure contour plot in Scenario 3. The pressure value throughout the model is almost uniform throughout the model, at the pressure value of

125 Pa. Reducing the available drifts resulted in less volume space for air to move around, thus making the overall pressure throughout the model averaging near to 125 Pa.

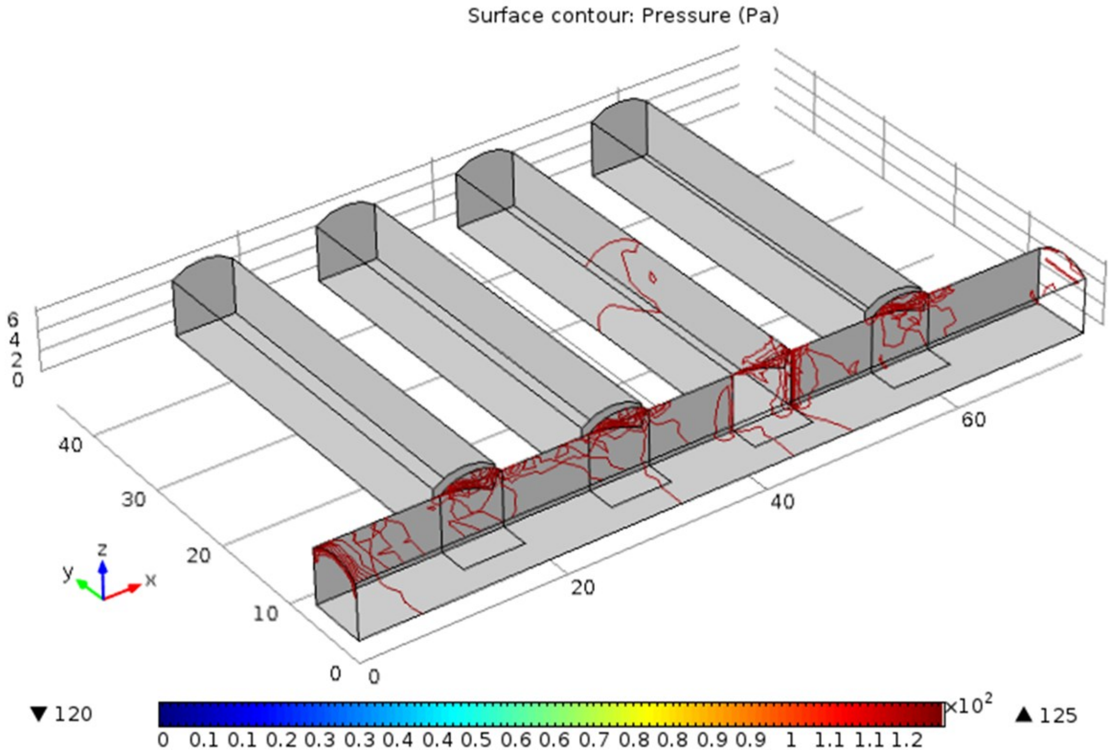


Figure 39. Surface contour of the pressure quantity (Pa) in the model of Scenario 3 (all drifts were blocked with air brattice, except at Drift 3).

4.2 Scheduling optimization

4.2.1 Error issues

During the debugging process by using the initial assumed data (20 workfaces with 5 working procedures), there had been several error results that were obtained during the processing. Instead of producing a Gantt chart, the program could not complete the processing. These errors could be seen in the text with red font in Figure 40. The error in Figure 40 occurred after selecting the option to create a Gantt chart schedule based on workface priority. This error similarly happened to other errors when selecting the option to create schedule based on workface dependency, sharing machines, and workface sorting. It was found out later that the cause of this error is the location file of the input data was in an incorrect folder directory.

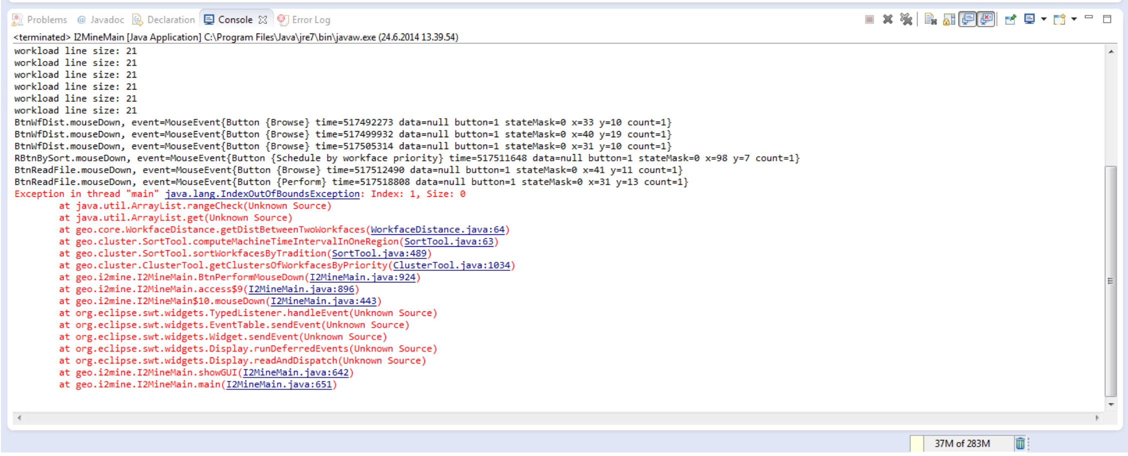


Figure 40. Error message after processing the input data, in order to find the Gantt chart for scheduling based on workface priority.

Another problem that was found is the processing that sometimes took a long time, due to not enough memory for the program. This happened when finding the optimized Gantt chart for the scheduling based on only one machine set. The reason this high time consumption is that it required more permutations calculation. When this happened, the program just went into a busy mode condition without any response.

During the processing, another error was that when the total amount of workface was changed into an amount different than 20 (20 workfaces was the default setting), the program could not process any of the input data. The solution for this problem was by updating the program code manually, in order to make the program working properly.

Another error was that the scheduling optimization software gave a total time to finish the whole 35 workfaces in less than 1 second in the Gantt chart. This is too fast for the mining operation to complete the drifting of 35 workfaces. Trial and error by changing the unit of the operating speed of the equipment then gave more reasonable results.

It was also often found that when the program needs a long time to process several type of scheduling, such as based on workface priority or workface dependency. The program took a long processing time to process and did not produce any Gantt chart. Instead, it only gave an error message. The scheduling of LHD and truck also has bug in the time formatting, which did not give clear time unit of the time results.

4.2.2 The results of the weekly plan of Kittilä mine

After the Kittilä weekly mine plan were inputted into the program as input data, the program was first executed to find the scheduling based on the priority of each workface. The priority of each workface was equally set to the value of “1” just for simplicity. Actually, were more than 3 workfaces in the input data, while the program only gave options of 3 priority levels.

After the program started to run, it could not produce any Gantt chart and it only gave an error message stating out of heap memory. It seems that this required a lot of computation and there might be a loop in the programming. The same thing happened with scheduling by workface dependency.

The Gantt chart produced based on scheduling by sharing machines can be seen in Figure 41. The Gantt chart shows that the whole mining process could be finished in 52 hours. It has to be noted that the Gantt chart does not include the change shift time, break time, lunch time and maintenance time. Therefore, the optimization program gives a very short timeline. The Gantt chart also displayed different number labels of the workface ID numbers compared to the original workface ID numbers in the input data. The software automatically subtracted all of the ID numbers in the Gantt chart by 1. For example, Workface 1 (original ID) becomes Workface 0 (Gantt chart), Workface 2 becomes Workface 1, and further on. This different display of workface identity number at the vertical axis graph of Figure 41was then corrected manually in Microsoft PowerPoint to make it easier to understand.

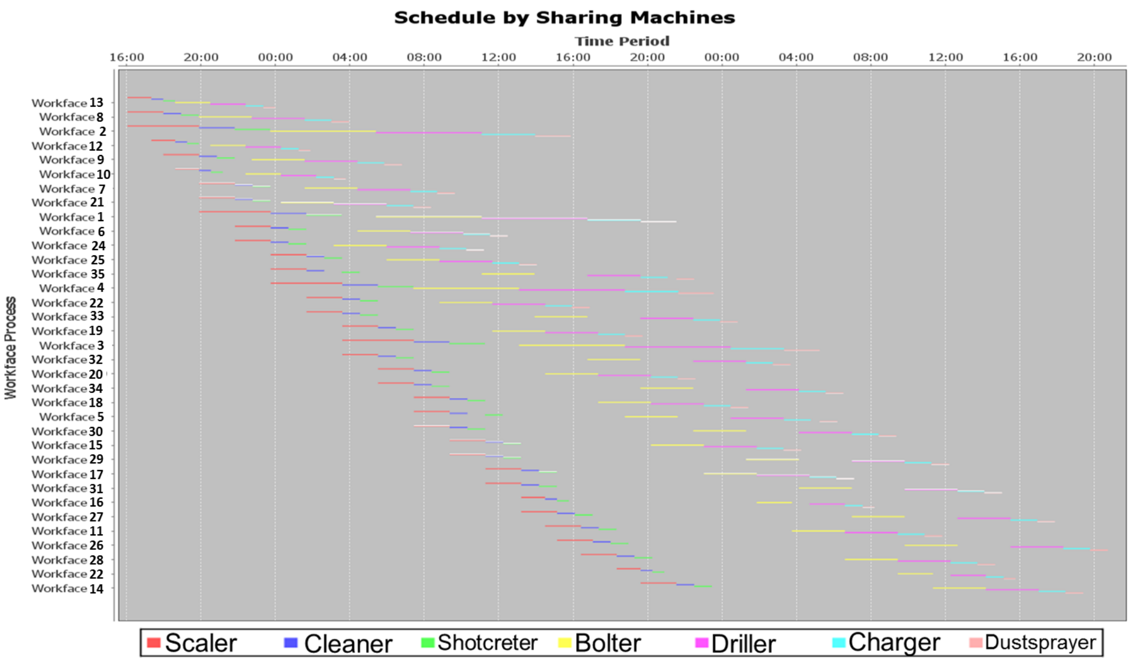


Figure 41. The Gantt chart for equipment schedule in the weekly plan of Kittilä based on machine set.

Figure 42 shows the Gantt chart resulted from the scheduling based on the workface sorting. It has the same formatting layout as in Figure 41. According to Figure 41, the scheduling based on sharing machines would need 2 days 4 hours and 40 minutes to complete the operation. The scheduling based on sorting workface in Figure 42 also produced a similar Gantt chart like in Figure 41 with the same amount of time to complete the whole workface.

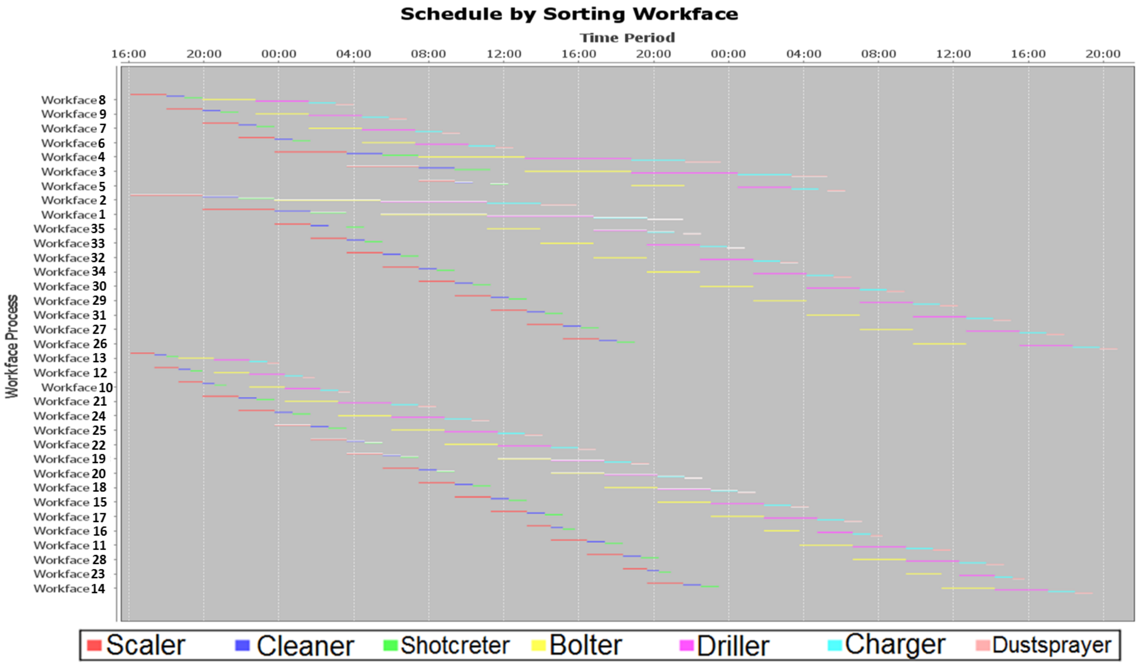


Figure 42. The Gantt chart based on sorting workface with using the input data from the weekly plan of Kittilä.

Although in both scheduling results (based on machine set and sorting workface) needed the same amount of time in the end, they have different orders for the equipment to start operating. Regarding the Gantt charts, it is odd that both of those scheduling types suggested three workfaces to be started in the same time, while in the input data, there were only two machines of scaler for "Procedure 0". This also occurred for other workfaces in the rest of the Gantt chart.

The result of the scheduling optimization of LHD and truck can be seen in Table 6. The program gave an order of which workface has to be mined first. The "start time" and "end time" column results have bug display issue due to the format in the programing. The "Workload Done" column explains the amount of material that had to be removed from the workface. The "Overflow" column is about whether if there is still blasted material left at that workface for loading in the next round or not.

Table 6. Scheduling of the LHD and truck allocation based on the scheduling optimization program.

Work-face order	Work-face ID	Start Time	End Time	Duration (hour)	Truck Number	Workload Done (ton)	Work Left	Over-flow	Level
1st	13	1.41E+12	1.41E+12	42.5	1	8500	0	null	1
2nd	12	1.41E+12	1.41E+12	42.0	1	8500	0	null	2
3rd	10	1.41E+12	1.41E+12	42.0	1	8500	0	null	3
4th	8	1.41E+12	1.41E+12	37.0	1	8500	0	null	4
5th	9	1.41E+12	1.41E+12	36.7	1	8500	0	null	5
6th	21	1.41E+12	1.41E+12	2.4	1	450	0	null	6
7th	7	1.41E+12	1.41E+12	33.8	1	8500	0	null	7
8th	24	1.41E+12	1.41E+12	24.5	2	8500	0	null	8
9th	6	1.41E+12	1.41E+12	1.9	1	450	0	null	9
10th	25	1.41E+12	1.41E+12	48.8	1	8500	0	null	10
11th	2	1.41E+12	1.41E+12	3.4	1	450	0	null	11
12th	22	1.41E+12	1.41E+12	2.4	1	450	0	null	12
13th	19	1.41E+12	1.41E+12	45.8	1	8500	0	null	13
14th	1	1.41E+12	1.41E+12	3.6	1	450	0	null	14
15th	35	1.41E+12	1.41E+12	2.9	1	450	0	null	15
16th	20	1.41E+12	1.41E+12	21.2	2	8500	0	null	16
17th	4	1.41E+12	1.41E+12	33.1	1	8500	0	null	17
18th	33	1.41E+12	1.41E+12	51.3	1	8500	0	null	18
19th	18	1.41E+12	1.41E+12	45.6	1	8500	0	null	19
20th	32	1.41E+12	1.41E+12	52.3	1	8500	0	null	20
21st	15	1.41E+12	1.41E+12	1.1	2	450	0	null	21
22nd	3	1.41E+12	1.41E+12	34.3	1	8500	0	null	22
23rd	5	1.41E+12	1.41E+12	36.4	1	8500	0	null	23
24th	34	1.41E+12	1.41E+12	2.8	1	450	0	null	24
25th	17	1.41E+12	1.41E+12	45.5	1	8500	0	null	25
26th	16	1.41E+12	1.41E+12	45.7	1	8500	0	null	26
27th	30	1.41E+12	1.41E+12	49.3	1	8500	0	null	27
28th	11	1.41E+12	1.41E+12	21.0	2	8500	0	null	28
29th	29	1.41E+12	1.41E+12	51.1	1	8500	0	null	29
30th	28	1.41E+12	1.41E+12	2.6	1	450	0	null	30
31st	31	1.41E+12	1.41E+12	2.6	1	450	0	null	31
32nd	23	1.41E+12	1.41E+12	49.2	1	8500	0	null	32
33rd	27	1.41E+12	1.41E+12	47.7	1	8500	0	null	33
34th	14	1.41E+12	1.41E+12	1.1	2	450	0	null	34
35th	26	1.41E+12	1.41E+12	48.2	1	8500	0	null	35

The monthly report of Kittilä's actual drifting activity during September 2013 for those 35 workfaces (that were simulated in this research) can be seen in Table 7. The table shows the progress that was made by the operation department specifically during 4 September 2013 until 10 September 2013. From the 35 workfaces that were planned for mining during that period, only 16 workfaces underwent drifting activity. This means only 50% of the planned workfaces were realised. While based on the Gantt charts produced by the scheduling optimization software, those 35 workfaces could be finished in 2 days 4 hours and 40 minutes. This is a big difference between the scheduling by the optimization software and the actual reality that happened. This is because the scheduling optimization software did not take into account the time for maintenance, break time, and change shift time. By the end of September 2013, all of the planned workfaces in Kittilä's Weekly Plan 4-10 September 2013 had already been drifted in Kittilä mine.

Table 7. Actual realised drifting activity at the 35 planned workfaces during September 2013 at Kittilä mine (Agnico Eagle, 2014c).

Workface ID number	Workface name	Date																														Drifting round	Fan pattern drilling	Fixing the tunnel	Meter	Monthly Target	Planned For 4-10/9/2013	Acutal realisation, 4-10/9/2013	Actual realisation, 1-30/9/2013
		S	M	T	W	T	F	S	S	M	T	W	T	F	S	S	M	T	W	T	F	S	S	M	T	W	T	F	S	S	M								
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30								
1	VT-TUT												1		1				1		1		1		1			1		1		8	0	1	37.6	70	2	×	✓
2	VT1				1			1		1			1								1			1			1					7	0	3	32.9	40	2	✓	✓
3	200P134			1							1				1									1					1			5	0	0	23.5	25	2	50%	✓
4	200P140						1					1			1									1								4	0	0	18.8	16	2	✓	✓
5	225P122			1			1																									2	0	0	9.4	15	1	✓	✓
6	225TP2			1			1									1																3	0	0	14.1	25	1	✓	✓
7	250P137	1			1																											2	0	2	9.4	0	1	✓	✓
8	250P122N																1															1	0	0	4.7	5	1	×	✓
9	250PP120S		1			1											1															3	0	0	14.1	20	1	✓	✓
10	375P153																										1					1	1	0	7.7	10	Fan drilling	×	✓
11	375P154															1																1	0	0	4.7	5	1	×	✓
12	375P155																										1					1	1	0	7.7	0	Fan drilling	×	✓
13	375P159																													1		1	1	0	7.7	0	Fan drilling	×	✓
14	375TP1																				1				1					1		3	0	0	14.1	15	1	×	✓
15	375TP2									1					1									1							1	4	0	1	18.8	20	1	✓	✓
16	400P135																										1					1	0	0	7.7	15	Fan drilling	×	✓
17	400P136	1																														1	1	0	4.7	15	1	×	✓
18	400P154	1						1								1								1							1	5	0	0	23.5	15	1	✓	✓
19	400P158			1				1						1				1				1										5	0	0	23.5	20	1	✓	✓
20	400P159			1				1						1				1				1				1					1	7	0	0	32.9	20	1	✓	✓
21	400TP1			1						1			1						1			1				1					1	7	0	0	32.9	20	1	✓	✓
22	400TP2				1									1											1							3	0	0	14.1	20	1	✓	✓
23	425P139																											1				1	0	0	4.7	15	Fan drilling	×	✓
24	425P140																	1												1		2	0	0	9.4	5	1	×	✓
25	425P141																	1														1	0	0	4.7	5	1	×	✓
26	425P145							1										1					1									3	0	0	14.1	10	1	✓	✓
27	425P149					1									1								1									3	0	0	14.1	15	1	✓	✓
28	425TP2											1			1							1				1						4	0	0	18.8	25	1	×	✓
29	450PP137S	1																1							1							3	0	0	14.1	20	1	×	✓
30	450PP150S																															0	0	0	0	15	1	×	✓
31	450TP1				1				1				1										1					1				5	0	0	23.5	20	1	✓	✓
32	475P131												1									1						1				3	0	0	14.1	15	1	×	✓
33	475PP137S																														1	1	0	0	4.7	20	1	×	✓
34	475TP1		1													1						1										3	0	0	14.1	20	1	×	✓
35	500TP1		1						1				1			1				1			1					1				7	0	0	32.9	25	1	✓	✓
																																					✓	16	35
																																					×	18	0
																																					50%	1	0

Explanation: drifting one round; fixing the drift; heightening the tunnel; heightening the drift+drifting; fan drilling pattern

4.2.3 The results of the scenarios

After the new scenarios were created, every input data of each scenario were inputted into the optimization software and processed one by one to obtain the Gantt charts and the LHD and truck scheduling. They all have the same layout as the Gantt charts in Sub-chapter 4.2.2. The summary of all Gantt chart per scenario can be seen below in Table 8. Scenario 11 has the shortest time to complete, which is 1 day, 12 hours and 35 minutes. The longest one to complete is Scenario 4 (2 days, 18 hours and 49 minutes). Scenario 11 has the shortest time because it only had 18 workfaces, while Scenario 4 had 35 workfaces with only 2 machines sets available.

Table 8. Summary of the order of which workface has to be mined first for each scenario and the total required time to mineout all of the workface based on machine set scheduling.

Work face order	Kittilä weekly plan	Scenario 1	Scenario 2	Scenario 3	Scenario 4	Scenario 5	Scenario 6	Scenario 7	Scenario 8	Scenario 9	Scenario 10	Scenario 11
1st	13	10	13	13	8	8	13	13	12	6	9	5
2nd	8	6	8	8	32	14	8	8	7	11	29	18
3rd	2	2	2	2	9	2	2	2	2	2	7	2
4th	12	9	12	12	33	9	12	12	10	10	30	17
5th	9	4	9	9	7	13	9	9	21	4	8	3
6th	10	7	10	10	31	23	10	10	8	20	28	1
7th	7	18	7	7	29	7	7	7	24	1	12	16
8th	21	1	21	21	21	1	21	21	9	25	26	13
9th	1	3	1	1	30	28	1	1	1	3	10	4
10th	6	21	6	6	19	6	6	6	25	8	11	11
11th	24	22	24	24	18	11	24	24	15	32	27	9
12th	25	32	25	25	25	35	25	25	4	9	15	12
13th	35	19	35	35	26	4	35	35	17	21	22	6
14th	4	5	4	4	10	12	4	4	35	5	17	7
15th	22	30	22	22	11	24	22	22	18	30	23	14
16th	33	16	33	33	27	33	33	33	3	29	18	8
17th	19	29	19	19	15	3	19	19	33	18	24	15
18th	3	17	3	3	24	32	3	3	22	13	16	10
19th	32	31	32	32	12	21	32	32	32	31	21	-
20th	20	15	20	20	28	16	20	20	19	14	14	-
21st	34	27	34	34	14	34	34	34	6	27	25	-
22nd	18	12	18	18	23	17	18	18	34	19	13	-
23rd	5	26	5	5	20	5	5	5	20	26	20	-
24th	30	14	30	30	2	30	30	30	5	15	2	-
25th	15	28	15	15	13	22	15	15	30	28	2	-
26th	29	13	29	29	3	29	29	29	29	16	4	-
27th	17	24	17	17	1	18	17	17	16	24	1	-
28th	31	8	31	31	4	31	31	31	31	17	6	-
29th	16	23	16	16	35	19	16	16	11	23	32	-
30th	27	25	27	27	34	27	27	27	27	12	5	-
31st	11	20	11	11	6	20	11	11	13	22	31	-
32nd	26	11	26	26	22	26	26	26	28	7	19	-
33rd	28	-	28	28	5	15	28	28	26	-	-	-
34th	23	-	23	23	16	25	23	23	23	-	-	-
35th	14	-	14	14	17	10	14	14	14	-	-	-
Start time	12:48	14:00	16:10	16:35	16:45	17:22	17:29	17:33	17:37	17:41	17:45	8:25
Finish time	17:28	18:40	20:54	21:15	11:34	22:02	22:09	22:13	19:27	22:21	7:51	21:00
Total time	2 days 4 hours 40 minutes	2 days 4 hours 40 minutes	2 days 4 hours 44 minutes	2 days 4 hours 44 minutes	2 days 18 hours 49 minutes	2 days 4 hours 40 minutes	2 days 4 hours 40 minutes	2 days 4 hours 40 minutes	2 days 1 hours 50 minutes	2 days 4 hours 40 minutes	2 days 14 hours 06 minutes	1 day 12 hours 35 minutes

Table 9 (page 61) shows the summary of the scheduling based on workface sorting from all scenarios. When compared to Table 8, the total time needed for completing every scenario is the same value. The only difference in every scenario is about the workface order and which workface has to be mined first. In Table 8, the mining starts first at Workface 13, while in Table 9 most of the operation starts first at Workface 8.

Table 9. Summary of the order of which workface has to be mined first for each scenario and the total required time to mineout all of the workface based on workface sorting scheduling.

Work face order	Kittilä weekly plan	Scenario 1	Scenario 2	Scenario 3	Scenario 4	Scenario 5	Scenario 6	Scenario 7	Scenario 8	Scenario 9	Scenario 10	Scenario 11
1st	8	6	8	8	32	8	8	8	7	6	29	2
2nd	9	4	9	9	33	9	9	9	8	4	30	1
3rd	7	3	7	7	31	7	7	7	9	3	28	5
4th	6	5	6	6	29	6	6	6	4	5	26	3
5th	4	2	4	4	30	4	4	4	3	2	27	4
6th	3	1	3	3	25	3	3	3	6	1	22	9
7th	5	32	5	5	26	5	5	5	5	32	23	6
8th	2	30	2	2	27	2	2	2	2	30	24	7
9th	1	29	1	1	24	1	1	1	1	29	21	8
10th	35	31	35	35	28	35	35	35	35	31	25	18
11th	33	27	33	33	23	33	33	33	33	27	20	17
12th	32	26	32	32	2	32	32	32	32	26	2	16
13th	34	28	34	34	1	34	34	34	34	28	1	13
14th	30	24	30	30	35	30	30	30	30	24	32	11
15th	29	23	29	29	34	29	29	29	29	23	31	12
16th	31	10	31	31	22	31	31	31	31	11	19	14
17th	27	9	27	27	16	27	27	27	27	10	9	15
18th	26	7	26	26	17	26	26	26	26	20	7	10
19th	13	18	13	13	8	14	13	13	12	25	8	-
20th	12	21	12	12	9	13	12	12	10	8	12	-
21st	10	22	10	10	7	23	10	10	21	9	10	-
22nd	21	19	21	21	21	28	21	21	24	21	11	-
23rd	24	16	24	24	19	11	24	24	25	18	15	-
24th	25	17	25	25	18	12	25	25	15	13	17	-
25th	22	15	22	22	10	24	22	22	17	14	18	-
26th	19	12	19	19	11	21	19	19	18	19	16	-
27th	20	14	20	20	15	16	20	20	22	15	14	-
28th	18	13	18	18	12	17	18	18	19	16	13	-
29th	15	8	15	15	14	22	15	15	20	17	3	-
30th	17	25	17	17	20	18	17	17	16	12	4	-
31st	16	20	16	16	13	19	16	16	11	22	6	-
32nd	11	11	11	11	3	20	11	11	13	7	5	-
33rd	28	-	28	28	4	15	28	28	28	-	-	-
34th	23	-	23	23	6	25	23	23	23	-	-	-
35th	14	-	14	14	5	10	14	14	14	-	-	-
Start time	13:00	16:07	16:31	16:39	16:47	17:24	17:30	17:35	17:39	17:43	17:46	8:25
Finish time	17:40	20:47	21:11	21:11	11:37	22:04	22:10	22:15	19:28	22:23	7:52	21:01
Total time	2 days 4 hours 40 minutes	2 days 4 hours 40 minutes	2 days 4 hours 40 minutes	2 days 4 hours 32 minutes	2 days 18 hours 50 minutes	2 days 4 hours 40 minutes	2 days 4 hours 40 minutes	2 days 4 hours 40 minutes	2 days 4 hours 49 minutes	2 days 4 hours 40 minutes	2 days 14 hours 6 minutes	1 day 12 hours 36 minutes

5 Discussion

5.1 Mine ventilation modelling

Based on the results, there were some differences between Scenario 1 (all drifts are open), Scenario 2 (all drifts are blocked with LHD except at Drift 3) and Scenario 3 (all drifts are blocked with air brattice except at Drift 3). According to the result from the velocity contour plot, velocity at the centre part of the drift in Scenario 1 could still achieve 0.5 m/s while in Scenario 3, it only reached 0.3 m/s. Overall, it could be said that the airflow that went into every drift is very low, only around 0.4 m/s.

In every model of each scenario, the average velocity magnitude of the whole surface, at the end surface at every drift and at the main access tunnel was also obtained. It was not possible to obtain the average value of the velocity in a partial volume of the model in COMSOL. The only possibility was to obtain the average value of the partial surface of the whole model. Therefore, during investigating the average velocity magnitude in each drift, it was done only for the value at the surface; not in the volume or space of each drift.

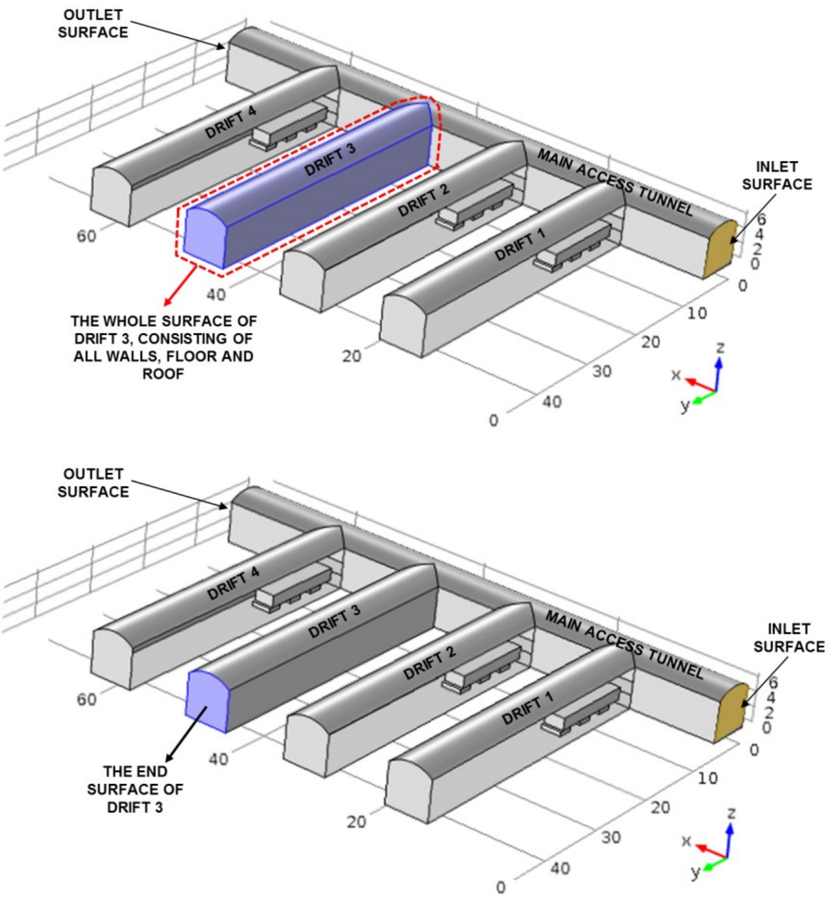


Figure 43. The surface area that was selected in Drift 3, in order to obtain the average airflow velocity; the values are compiled in Table 10.

Figure 43 illustrates the definition of "the whole surface" and "the end surface" at each drift. In Figure 43, Drift 3 was taken as an example and the same principle applies to other drifts. The surfaces that were studied from the main access tunnel are the roof, floor, and wall of it. These data were compiled together and can be seen in Table 10.

Table 10. Average airflow velocity magnitude at the whole surface and at the end of every drift for each scenario.

Scenario Type	Average velocity magnitude at the whole surface of Drift (m/s)				Average velocity magnitude at the end surface of Drift (m/s)				Average velocity magnitude at the main access tunnel (m/s)
	1	2	3	4	1	2	3	4	
Scenario 1	0.33	0.30	0.30	0.24	0.08	0.23	0.21	0.19	2.47
Scenario 2	0.27	0.23	0.30	0.21	0.04	0.13	0.12	0.12	2.94
Scenario 3	0	0	0.31	0	0	0	0.08	0	2.49

The scenario that has the highest average velocity magnitude at the main access tunnel is in Scenario 2 (2.94 m/s). This high velocity is related with the air turbulence and friction for airflow to enter the drift due to the presence of LHD's at Drift 1, 2, and 4. The inlet air pressure had to overcome to this turbulence by giving higher pressure, thus resulting in higher velocity magnitude at the main access tunnel (this higher pressure could be seen Figure 38 about pressure contour plot and later on in Table 11).

To easily understand the velocity trend in Table 10, Figure 44 and Figure 45 clearly illustrate the graphs for each scenario. In Figure 44, Scenario 1 has the highest average airflow velocity in the whole surface of every drift, continued by Scenario 2 and Scenario 3. Figure 45 also has similar finding to Figure 44, where Scenario 1 has the highest average airflow velocity in the end surface of every drift.

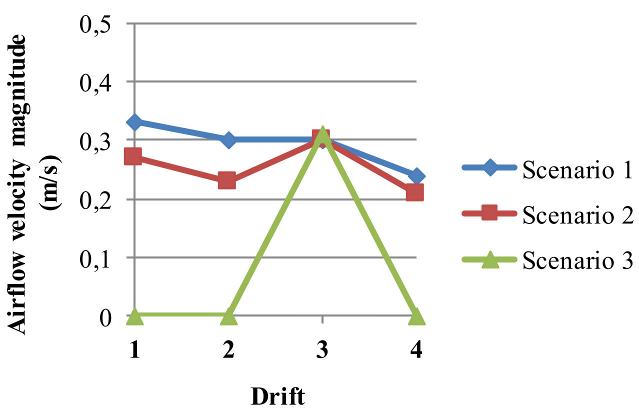


Figure 44. The graph comparing the average airflow velocity magnitude at the whole surface of each drift in all scenarios.

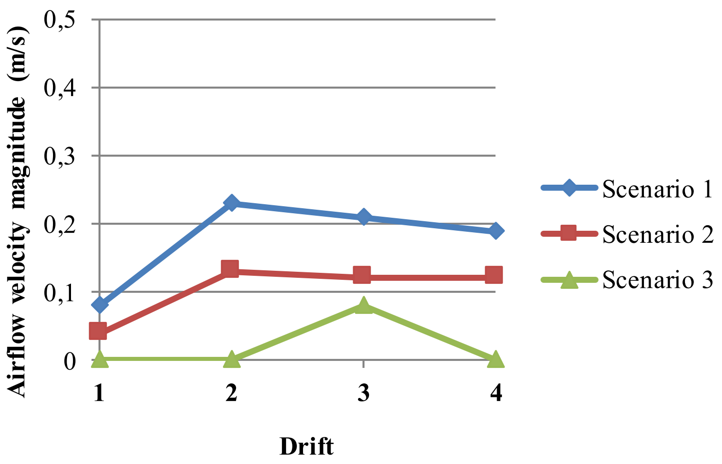


Figure 45. The graph comparing the average airflow velocity magnitude at the end surface of each drift in all scenarios.

In general, Scenario 1 has the highest velocity for every drift and at the end of the drift, followed by Scenario 2 and Scenario 3 respectively. By looking into the result about the streamline plot, it could be seen that the airflow just mainly stays in the main access tunnel. Only few of the airflow went into the drifts. Scenario 1 has the highest average value of all drifts because all of the entrances of the drifts are freely open for the airflow to come in. Scenario 2 has LHD's blocking the entrances, thus increasing the resistance for air to flow into the drifts.

In the case of Scenario 1, the drift closest to the inlet (Drift 1) has the highest average velocity at the surface. For Scenario 2, the drift that is unblocked (Drift 3) has the highest average velocity at the surface. Scenario 3 is the same as in Scenario 2, where it is obvious that only Drift 3 has airflow velocity in it. The low results of airflow velocity magnitude in the end surface of Drift 1 of Scenario 1 and 2 mean that most of the airflow did not go to the end of the drift. The air only went through the entrance and then had turbulence. This turbulence would hinder the airflow to go to the end of the drift. This can be seen in the streamline plotting results (Sub-chapter 4.1.3), where Drift 1 has a lot of thick streamline which is proportional to the turbulent viscosity. The case is different in Drift 2 because the airflow has more momentum to flow to the end of the tunnel because there is less turbulent viscosity.

The proximity of Drift 1 to the inlet would be the reason of this higher turbulence in Drift 1. When the airflow is passing the intersection between Drift 1 and the main access tunnel, the turbulence is caused by the interaction between the air and the corners of the intersection. The resistance for air to enter Drift 1 means that the air would have higher tendency to remain in the main access tunnel.

At Drift 3 Scenario 3, the average velocity for the whole surface is at the same number as in other scenarios (0.3 m/s). However, the tendency for airflow to remain in the main access tunnel is also higher than in Scenario 1 and 2 because before the air arrived at the entrance of Drift 3, the air had just been traveling in a straight motion in the main access tunnel for 41 meters (from the inlet surface). Some air would enter Drift 3, but this air would become turbulence because Drift 3 was the first branch of the main access tunnel. This is the same situation as in Drift 1 Scenario 1, where Drift 1 was the first branch of the main access tunnel for airflow in Scenario 1. Therefore, the airflow that went into Drift 3 of Scenario 3 was the least than in Scenario 1 and Scenario 2.

All of the scenarios did not produce enough airflow velocity to dilute the gas particle at the end of Drift 3. In all of the scenarios at Drift 3 (which was the main subject of investigation because it was assumed that there was a blasting in Drift 3), they have an average velocity magnitude of less than 0.3 m/s. This is not sufficient enough compared to the minimum airflow requirement at Pyhäsalmi mine and the German mining regulation.

There was no specific type of installation at the intersection between Drift 3 and the main access tunnel which could maximize the airflow direction into Drift 3. This can be seen by how the main airflow from the inlet just still remains in the main access tunnel. Besides of that, in all of the drifts, there is no separator or a divider between the fresh airflow from the main access tunnel and return airflow from the end surface of the drifts. These two airflows have different directions. Collision between both of these opposing airflows would only create turbulence and reduce the resultant of the airflow velocity in the drifts.

The velocity of airflow in the main access tunnel would be different in the model if it had a rough and blocky surface as in real underground mines. There would be more friction and turbulence in the case of rough surface. Since the surface roughness of this model was using the default settings of the software (wall functions option), the airflow behaviour flows like a fluid flowing in a normal pipe. It could be noticed in the vertical slice plot where there is gradation of velocity that became lower when it is nearer to the wall. This gradation illustrates that the airflow was affected by the surface roughness of the wall in this research.

In the COMSOL simulation, it was possible to obtain the required pressure at the inlet for each scenario. This inlet pressure was needed to overcome the assumed pressure at the outlet surface, which was 120 Pa. The inlet pressure has to be higher than 120 Pa so air could flow from the inlet to the outlet. Subtracting the pressure value at the inlet to the pressure value at the outlet would provide the value of pressure drop in each scenario. The results of the average pressure at the inlet surface and the pressure drop could be seen in Table 11. Scenario 2 gave the highest value of pressure drop because Scenario 2 has the presence of LHD's in the entrance of Drift 1, 2, and 4. These LHD's created additional air friction and resistances to the ventilation system, thus requiring more inlet pressure to sustain the 4 m/s velocity at the inlet and to overcome those resistances.

Table 11. The average pressure at the inlet and outlet and the pressure drop for each scenario.

Scenario Type	Average pressure at inlet (Pa)	Average pressure at outlet (Pa)	Pressure drop (Pa)	Pressure drop (kPa)
Scenario 1	123.82	120.08	3.74	0.00374
Scenario 2	123.93	120.03	3.90	0.00390
Scenario 3	122.42	120.05	2.37	0.00237

By knowing the pressure drop data, thus the required air power, fan motor power and total cost for each scenario could be obtained. This can be seen in Table 12. Since scenario 1 and Scenario 2 did not use air brattice, thus the installation cost is zero. On the other hand, the total cost for installing the air brattice to block Drift 1, 2 and 4 in Scenario 3 would be 60 euro (the process to get this value was explained in Sub-chapter 3.1.6). Scenario 3 gave the cheapest option, followed by Scenario 1 and Scenario 2 respectively. It has to be considered that although Scenario 2 did not need additional cost for installing air brattice, but parking an LHD in less than 100 meter radius area has to be in line with the mine safety regulation about the safety distance for equipment from the blasting point. Probably there are some underground mines that have higher risk for spalling rock during blasting compared to other mines and have a larger safe distance area during blasting. Therefore, it would not be possible for these mines to park an LHD within the vicinity of the blasting point.

Table 12. The air power of the fan and the total cost for each scenario in one year.

Scenario Type	Air Power (kW)	Required Fan Motor Power (kW)	Operating cost of the fan per year (EUR)	Installation cost in each scenario (EUR)	Total cost in one year (EUR)
Scenario 1	0.558	0.744	596	0	596
Scenario 2	0.582	0.775	622	0	622
Scenario 3	0.353	0.471	378	60	437

Although economically Scenario 3 has the cheapest cost, but it is technically not realistic to apply in the underground mine because it did not produce enough air velocity to dilute the gaseous particles from blasting. The practice of blocking the drifts' entrance with LHD's and air brattice have showed that it would not increase the airflow into the drift that was not blocked at all. There has to be another installation such as an additional air duct or installing an air brattice that would direct the wind from the main access tunnel into the drift.

Several additional models were made to experiment this idea. These models could be seen in Figure 46 and Figure 47. The basic layout and geometry was taken from the model of Scenario 2 (drifts blocked with LHD). There were eight different configurations of air brattices located at the entrance of Drift 3. Three types were in an L-shaped and five types were in a straight line. There were variations in the location of the air brattice to the left hand side wall. Most were having a length of 20 meters in Drift 3, while type 7 was extended 30 meters and type 8 was only 10 meters into Drift 3 (both type 7 and 8 are not necessary to be illustrated in this writing).

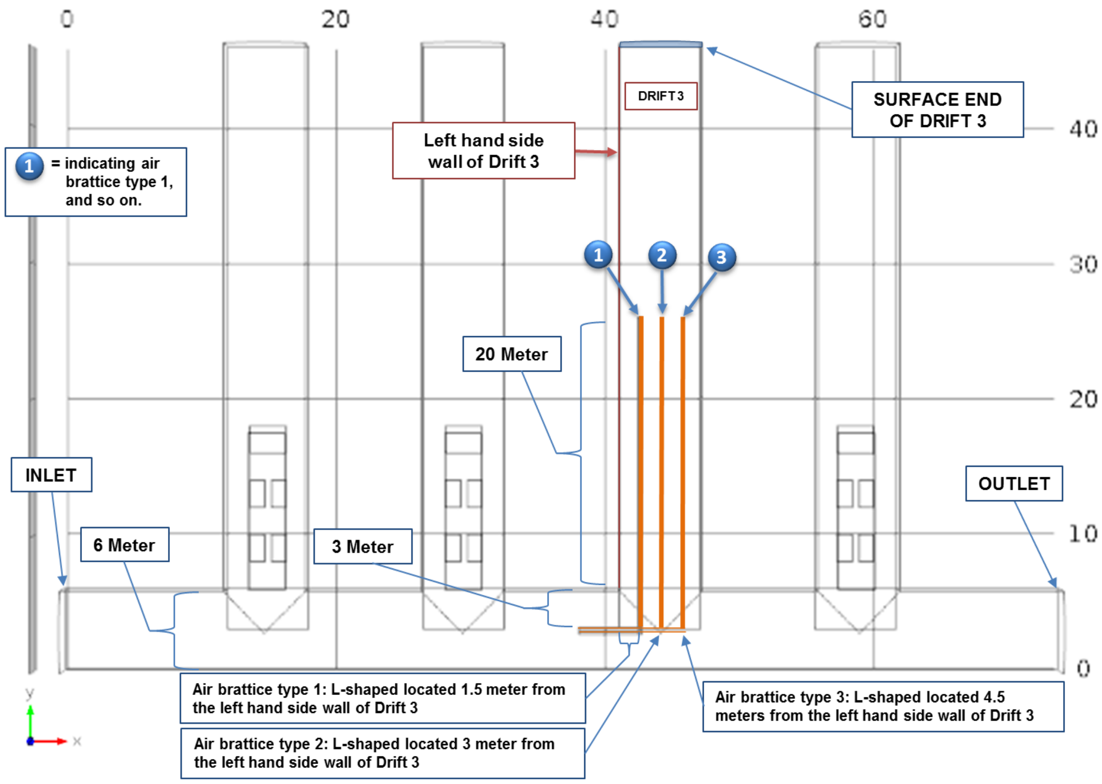


Figure 46. Illustration of the air brattice configuration type 1, 2 and 3 for directing air into Drift 3.

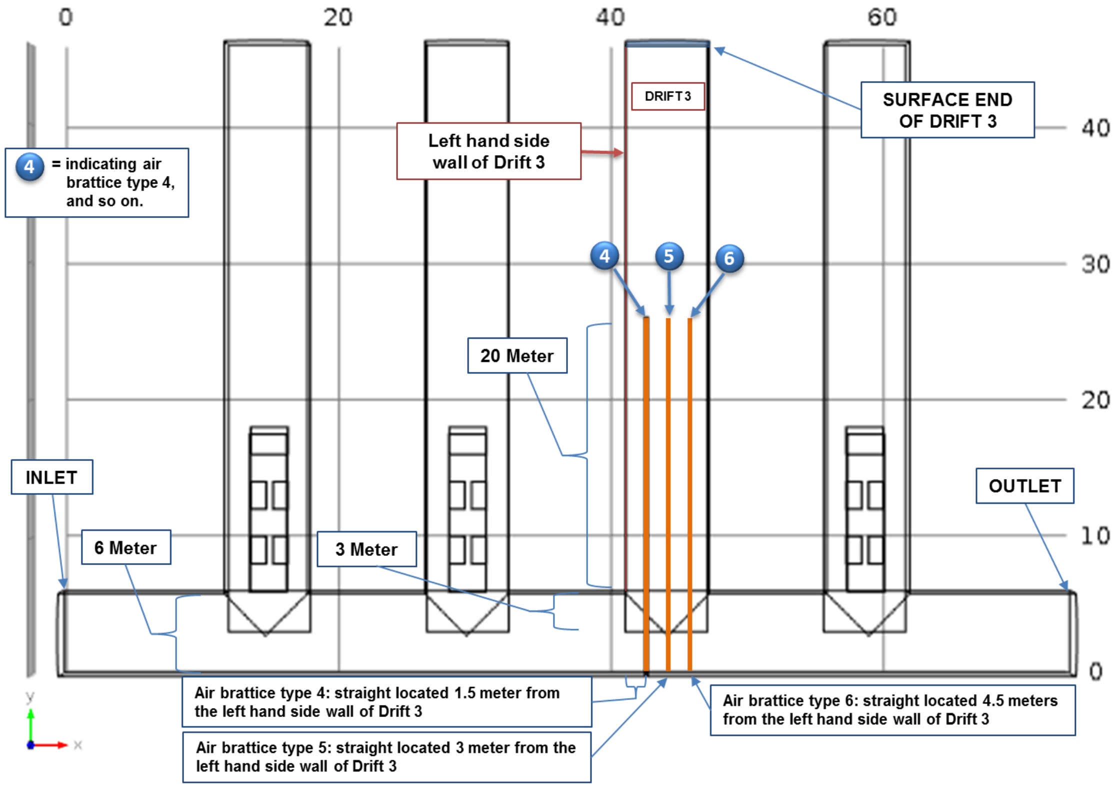


Figure 47. Illustration of the air brattice configuration type 4, 5, 6 for directing air into Drift 3.

The results of applying these air brattice configurations could be seen in Table 13 (page 69). Configuration type 4 gave the highest average air velocity at the whole surface of Drift 3 (7.08 m/s), while configuration type 7 gave the highest average air velocity at the surface end of Drift 3 (1.44 m/s). Every air brattice in a straight configuration are more reasonable to apply because the average air velocity at the surface end of Drift 3 is higher than the minimum requirement by the German mining law (0.5 m/s). However, if the Pyhäsalmi mine's airflow velocity requirement of 1 m/s was the only consideration, only air brattice type 4 would be the best applicable in this case due to its highest value of average velocity. In reality, all of these air configurations might not be realistic for application because its proximity to the blasting point and the time required for installing and uninstalling the air brattice. Also, these air brattice configurations neglected the possibility for mine equipment to pass through these area.

About the wall lift-off parameter, there were some areas in the original scenario model which has a wall lift-off value of more than 11.06. An example of this could be seen in Figure 48 (page 69), which depicts the case for Scenario 1. The same thing happened also in Scenario 2 and 3. It was noticed that the high wall lift-off level happened in areas where there was a lot of turbulence. These were around the inlet, the outlet and at the corners of every intersection. The wall lift-off in those areas mean that the results in those areas were not accurate enough and the size of the mesh should be made finer.

Table 13. The average air velocity magnitude at the surface and at the end of Drift 3 for each type of air brattice configuration.

Configuration type of the air brattice	Air brattice shape	Brattice section length in Drift 3 (m)	Distance to the left hand side wall when looking from the drift entrance (m)	Average air velocity magnitude at the surface of Drift 3 (m/s)	Minimum air velocity magnitude at the surface of Drift 3 (m/s)	Average air velocity at surface end of Drift 3 (m/s)
1	L shape	20	1.5	1.93	0	0.42
2	L shape	20	3	1.78	0.01	0.44
3	L shape	20	4.5	1.82	0	0.34
4	Straight	20	1.5	**7.08**	0.01	1.21
5	Straight	20	3	4.92	0.01	0.79
6	Straight	20	4.5	6.02	0.03	0.99
7	Straight	30	3	6.02	0.01	**1.44**
8	Straight	10	3	3.43	0.02	0.56

It was already been tried during the research to create the model by using finer meshes but on one testing, it took more than 3 days to compute and was still not finished, and some could not complete the calculation because the program said there was error. The real reason behind this error was not really clear. Therefore, it was decided for not continuing the modelling by using finer mesh size.

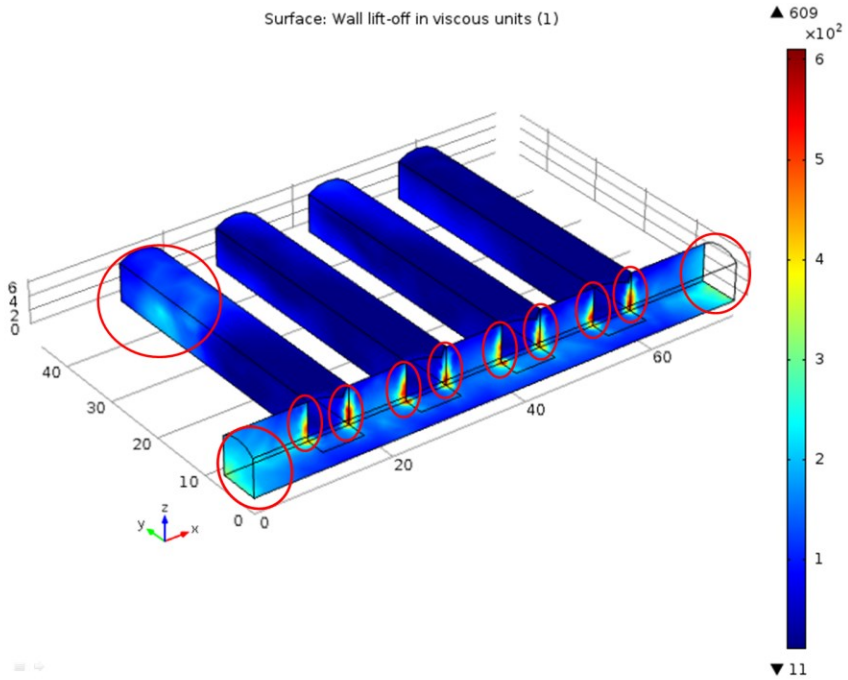


Figure 48. Several areas in Scenario 1 where the wall lift-off is more than 11.06, indicated by the red circles.

5.2 Scheduling optimization software

After trying and using this new program for scheduling optimization, there were a lot of issues for using this program. Several issues were already mentioned before in the sub-chapter about "Error issues" in Sub-chapter 4.2.1. While the other issues are related with how close is this scheduling optimization with the real actual condition during mining operation and its functionality.

In this scheduling optimization software, it did not recognize the possibility if the operation activity in the mine would work with different types of materials, such as various grade of ore and different type of waste. At Kittilä mine, they have 5 different stockpiles for ore, where each stockpile is designated for a certain type of ore grade. Besides of that, they also have waste dump specifically for NAF (Non-Acid Forming) and PAF (Potentially Acid Forming) material. There are also several intermediate stockpiles for backfilling the stopes in different locations in Kittilä's underground mine. These different stockpiles were also not accommodated in the software. During this research, all of the materials were just assumed to be the same type, in order to keep continuing with the testing of the scheduling optimization software.

During inputting the data related with the machine operating info from Kittilä mine, it was impossible to input the data of two different equipment brand in the same type of machine activity. For example, in Kittilä, they have two different types of equipment for the cleaning activity, one is the wheel loader and the other one is the excavator. Both have different operating and moving speed and these differences were also not accommodated in the scheduling optimization software.

This research used a total workface of 35 workfaces at most. The process to input all of the data from scratch into the scheduling optimization software required a tedious amount of work in preparing the data. All of the input data had to be first converted to become text files. The writing format in the text files required some kind of a particular order and matrixes. This means a lot of preparation had to be done before using the software. It could be imagined how much work and time would be needed if the underground mine has a larger scale of mining operation, with for example, 50 workfaces. It could be said that it is not time efficient in order to optimize a lot of workfaces by using this software.

This study only looked into only one weekly plan of Kittilä mine. There could be a further possibility to schedule the mine with a longer time horizon, such as for a monthly plan and a three month rolling plan. If the optimizing software would be used for a longer time range, this would mean more workfaces have to be prepared and this would take a long time of preparation (as in the case mentioned in the previous paragraph).

This software used input data that were based on generalisation, simplifying the complexity of the mine condition. While in reality, there are a lot of "black swan" events and a lot of uncertainties. There are a lot of factors that could influence the activities at the field. For example in the drifting activity, not all field condition for drifting has the same rock condition. Some workface could have worse rock condition than the other workface. Therefore, it would require more time activities in installing the needed rock support and reinforcement (rockbolt, shotcrete, wiremesh). Break time, change shift time, maintenance time, blasting time were also not included in the software's algorithm. It is still hard for this scheduling optimization software to create a schedule that would come very close to the actual condition at the field.

The software still has a basic user interface and the appearance still looks rigid compared to using marketable software. For example, when the user went back to the "read basic file" tab menu in the first step of processing the input data, the green check sign does not reset back into a red cross sign. The layout of the Gantt charts was not too comfortable and a little bit confusing to read for new users. The bar depicting each activity were to thin and it was hard to determine when should the activities start. The colour lines are also sometimes different for each line. Especially since the time line at the top horizontal axis always changes as the program starts a new scheduling process. The naming and numbering of the workface and equipment in the Gantt chart is also confusing because users always had to crosscheck it with the workface name in the input data. It would be better if there is an option for users to edit and change the name of the labels of the vertical and the horizontal axis manually.

For creating the scheduling based on machine set for all scenarios, the option of "three machine sets" was always selected. The reason for this is because there was a third machine fleet. But, this third machine fleet only had one cleaning equipment (Machine Type 2, this machine is like a small wheel loader and a small excavator for cleaning the floor of the drift from small boulders and uneven floor surface). Regarding this fact, the software thought that other equipment also existed in the third machine set. This could be seen in the Gantt charts where it gave time allocation for a third scaler, shotcreter, bolting machine, driller, charger, and service truck.

The software gave the shortest total time for Scenario 11 because it only had 18 workfaces. While the longest total time was for Scenario 4 which has 35 workfaces with only 2 machine sets to complete all of the workfaces. Reducing the amount of workface in the input data created fewer amounts of workload and task to complete for the operation department, therefore having shorter total time.

The software seems to provide the user with a schedule in an optimised way because in most the Gantt charts created by this program, they all have a short time scale to complete all workfaces (only need around 2 days in average). The software was able to group and order the workfaces into a certain pattern in each scenario. But a lot of factors were still not incorporated in the program. Besides of that, judging from the results between the scheduling "based on machine set" and "scheduling based on workface sorting", there is no difference in the amount of total time. This can be seen in Table 8 and Table 9 (page 60-61), where all of the total time between each scenarios in both type of scheduling are the same. This mean that either way of the scheduling type (based on machine set or based on workface sorting) would be just the same in this scheduling optimization software. This was caused by the algorithm that was used in both scheduling types were the same algorithm. Other types of scheduling which were based on priority and dependency were already tried but they always gave error message, stating that there was not enough memory to complete the computation.

6 Conclusion and Suggestions

6.1 Conclusion

The COMSOL software showed that all of those three scenarios (all drifts are open, some drifts are blocked with LHD, and some drifts are block with air brattice) are not sufficient enough to provide proper ventilation in Drift 3. Economically, Scenario 3 is the cheapest option, followed by Scenario 1 and then scenario 2. However, all of those scenarios are not reasonable to apply because they produced airflow velocity less than 0.5 m/s at the end of Drift 3. This is not enough regarding to the minimum airflow requirement in Pyhäsalmi mine or the German mining regulation.

In order to maximize the airflow at the end for Drift 3, there has to be an extra installation at the entrance of Drift 3. Several air brattice installations were simulated in this research to study this. It found out that the best air brattice layout for the case of Scenario 2 was air brattice configuration type 4 (straight-shaped air brattice, 20 meters length in Drift 3, located 1.5 meters to the left hand side wall of Drift 3) because it produced the highest average velocity of airflow at the surface of Drift 3 (7.08 m/s).

The scheduling optimization software has been able to group the workfaces from the input data. Besides of that, it was able to produce Gantt charts and the scheduling for LHD and dump trucks. The scheduling based on “machine set” and “workface sorting” gave the same amount of total time for every scenarios because it was using the same algorithm. This scheduling optimization software still lacks some aspects, such as:

1. Could not differentiate the type of materials and the specific dumping area.
2. Impossible to input two different types of equipment brand for the same activity.
3. Tedious amount of preparation for a large scale input data had to be done before using it.
4. Still has not incorporated break time, change shift time, maintenance time, and blasting time into the Gantt chart.
5. The Gantt chart and the LHD-truck scheduling results have a layout that is still confusing for new readers to understand.
6. There were bugs in the program and sometimes in certain cases it needed a lot of memory for processing.

6.2 Suggestions

There are several suggestions which could be considered to be done in the future and are related to this research. For the ventilation study in COMSOL, those suggestions are:

1. Explore the chance to investigate real problem case of underground mine ventilation in Finland by using COMSOL software. The modelling could be based on real data collected from the field (actual surface roughness of the tunnel, air characteristic, actual mine layout and geometry, and actual fan characteristics).

2. The complexity of the mine model that was used in this research could be increased. For example, this study was done in COMSOL using the "*Stationary Preset Studies*", so the future research could be done in the "*Time Dependent Preset Studies*", which is more useful to solve unsteady flow and pressure fields. Another example would be to modify the mine geometry in this research, or try to solve the problem of finer mesh size that was not possible to be done in this research.

3. The model that was used in this research could be recreated in a scale model in a ventilation laboratory. After that, the results obtained from the ventilation laboratory could be compared and analysed to the result from this research.

For the scheduling optimization software, the suggestions that could be considered are:

1. Before using the scheduling optimization software, all of the data that were needed to be used in the software were figured out manually. For example, were for finding the distances from one workface to another workface and the distance to the dumping point. If there is another software or program that could perform this function automatically, it would improve the efficiency before using the optimization software.

2. Other aspects that were not incorporated before in this software could be included to improve the accuracy of the results. These aspects are different types of materials in the underground mine, different types of dumping area, different types of equipment for one activity, and various activities of the mining operation (short break, lunch, time for change shift crew, blasting and ventilation time, and maintenance time).

3. Improve the layout and the readability of the Gantt chart and the LHD-truck scheduling result.
4. Solving the bugs and error that were found in the scheduling optimization software.
5. Look into an opportunity for making the software compatible with a dispatch and tracking system that is available on the market. This would integrate the software with actual conditions in the underground mine, thus creating a loop-feedback. This might improve the quality of the optimized schedule.
6. Trialling the software for a future mine plan (instead of only using past/historical mine plan), in order to see whether if the software is really optimizing the mining sequence or not.
7. Create a simple manual document that gives a detailed list of steps for new users to use the scheduling optimization software. This manual would be a valuable guide for new users to understand the usage of the software.

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Appendix

List of Appendices

Appendix 1. Input data of the weekly plan of Kittilä mine during 4-10 September 2013 into the scheduling optimization software

The distance from one drift to another drift.

		Drift name	VT-TUT	VT1	200P134	200P140	225P122	225TP2	250P137	250P122N	250PP120S	375P153	375P154	375P155	375P159	375TP1	375TP2	400P135	400P136	400P154	400P158	400P159	400TP1	400TP2	425P139	425P140	425P141	425P145	425P149	425TP2	450PP137S	450PP150S	450TP1	475P131	475PP137S	475TP1	500TP1
		Workface ID	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35
No.	Drift name	Workface ID																																			
1	VT-TUT	1	0.0	1272.8	4112.1	4014.8	4000.6	3995.8	3520.3	3772.3	3751.6	2468.4	2470.4	2470.0	2508.6	2512.7	2387.9	2382.9	2372.0	2470.4	2391.1	2380.7	2410.2	2366.6	2306.6	2295.5	2276.9	2230.8	1906.5	2326.5	2138.2	2001.2	2008.7	1884.9	1802.0	1972.3	1734.8
2	VT1	2	1272.8	0.0	3794.1	3696.8	3682.6	3677.8	3202.3	3454.3	3433.6	2150.4	2152.4	2152.0	2190.6	2194.7	2069.9	2064.9	2054.0	2060.8	2073.1	2062.7	2092.2	2048.6	1988.6	1977.5	1958.9	1912.8	1871.3	2008.5	1820.2	1683.2	1690.7	1566.9	1484.0	1654.3	1416.8
3	200P134	3	4112.1	3794.1	0.0	187.5	1593.5	1588.7	1386.2	1638.2	1617.5	2032.9	2034.9	2034.5	2073.1	2077.2	1952.4	2327.6	2316.7	2323.5	2335.8	2325.4	2354.9	2311.3	2605.7	2594.6	2576.0	2529.9	2488.4	2625.6	2756.7	2619.7	2627.3	2857.0	2774.1	2944.4	3096.9
4	200P140	4	4014.8	3696.8	187.5	0.0	1496.2	1491.4	1288.9	1540.9	1520.2	1935.6	1937.6	1937.2	1975.8	1979.9	1855.1	2230.3	2219.4	2226.2	2238.5	2228.1	2257.6	2214.0	2508.4	2497.3	2478.7	2432.6	2391.1	2528.3	2659.4	2522.4	2530.0	2759.7	2676.8	2847.1	2999.6
5	225P122	5	4000.6	3682.6	1593.5	1496.2	0.0	19.8	1274.7	1526.7	1506.0	1921.4	1923.4	1923.0	1961.6	1965.7	1840.9	2216.1	2205.2	2212.0	2224.3	2213.9	2243.4	2199.8	2494.2	2483.1	2464.5	2418.4	2376.9	2514.1	2645.2	2508.2	2515.7	2745.5	2662.6	2832.9	2985.4
6	225TP2	6	3995.8	3677.8	1588.7	1491.4	19.8	0.0	1269.9	1521.9	1501.2	1916.6	1918.6	1918.2	1956.8	1960.9	1836.1	2211.3	2200.4	2207.2	2219.5	2209.1	2238.6	2195.0	2489.4	2478.3	2459.7	2413.6	2372.1	2509.3	2640.4	2503.4	2510.9	2740.7	2657.8	2828.1	2980.6
7	250P137	7	3520.3	3202.3	1386.2	1288.9	1274.7	1269.9	0.0	331.6	310.9	1441.1	1443.1	1442.7	1481.3	1485.4	1360.6	1735.8	1724.9	1731.7	1744.0	1733.6	1763.1	1719.5	2013.9	2002.8	1984.2	1938.1	1896.6	2033.8	2164.9	2027.9	2035.4	2265.2	2182.3	2352.6	2505.1
8	250P122N	8	3772.3	3454.3	1638.2	1540.9	1526.7	1521.9	331.6	0.0	44.5	1693.1	1695.1	1694.7	1733.3	1737.4	1612.6	1987.8	1976.9	1983.7	1996.0	1985.6	2015.1	1971.5	2265.9	2254.8	2236.2	2190.1	2148.6	2285.8	2416.9	2279.9	2287.4	2517.2	2434.3	2604.6	2757.1
9	250PP120S	9	3751.6	3433.6	1617.5	1520.2	1506.0	1501.2	310.9	44.5	0.0	1672.4	1674.4	1674.0	1712.6	1716.7	1591.9	1967.1	1956.2	1963.0	1975.3	1964.9	1994.4	1950.8	2245.2	2234.1	2215.5	2169.4	2127.9	2265.1	2396.2	2259.2	2266.7	2496.5	2413.6	2583.9	2736.4
10	375P153	10	2468.4	2150.4	2032.9	1935.6	1921.4	1916.6	1441.1	1693.1	1672.4	0.0	26.4	26.0	132.8	136.9	273.3	683.9	673.0	679.8	692.1	681.7	711.2	667.6	962.0	950.9	932.3	886.2	844.7	981.9	1113.0	976.0	983.5	1213.3	1130.4	1300.7	1453.2
11	375P154	11	2470.4	2152.4	2034.9	1937.6	1923.4	1918.6	1443.1	1695.1	1674.4	26.4	0.0	28.0	134.8	138.9	275.3	685.9	675.0	681.8	694.1	683.7	713.2	669.6	964.0	952.9	934.3	888.2	846.7	983.9	1115.0	978.0	985.5	1215.3	1132.4	1302.7	1455.2
12	375P155	12	2470.0	2152.0	2034.5	1937.2	1923.0	1918.2	1442.7	1694.7	1674.0	26.0	28.0	0.0	134.4	138.5	274.9	685.5	674.6	681.4	693.7	683.3	712.8	669.2	963.6	952.5	933.9	887.8	846.3	983.5	1114.6	977.6	985.1	1214.9	1132.0	1302.3	1454.8
13	375P159	13	2508.6	2190.6	2073.1	1975.8	1961.6	1956.8	1481.3	1733.3	1712.6	132.8	134.8	134.4	0.0	177.1	313.5	724.1	713.2	720.0	732.3	721.9	751.4	707.8	1002.2	991.1	972.5	926.4	884.9	1022.1	1153.2	1016.2	1023.7	1253.5	1170.6	1340.9	1493.4
14	375TP1	14	2512.7	2194.7	2077.2	1979.9	1965.7	1960.9	1485.4	1737.4	1716.7	136.9	138.9	138.5	177.1	0.0	317.6	728.2	717.3	724.1	736.4	726.0	755.5	711.9	1006.3	995.2	976.6	930.5	889.0	1026.2	1157.3	1020.3	1027.8	1257.6	1174.7	1345.0	1497.5
15	375TP2	15	2387.9	2069.9	1952.4	1855.1	1840.9	1836.1	1360.6	1612.6	1591.9	273.3	275.3	274.9	313.5	317.6	0.0	603.4	592.5	599.3	611.6	601.2	630.7	587.1	881.5	870.4	851.8	805.7	764.2	901.4	1032.5	895.5	903.0	1132.8	1049.9	1220.2	1372.7
16	400P135	16	2382.9	2064.9	2327.6	2230.3	2216.1	2211.3	1735.8	1987.8	1967.1	683.9	685.9	685.5	724.1	728.2	603.4	0.0	36.1	442.4	454.7	444.3	473.8	7.7	876.5	865.4	846.8	800.7	759.2	896.4	1027.5	890.5	898.0	1127.8	1044.9	1215.2	1367.7
17	400P136	17	2372.0	2054.0	2316.7	2219.4	2205.2	2200.4	1724.9	1976.9	1956.2	673.0	675.0	674.6	713.2	717.3	592.5	36.1	0.0	431.5	443.8	433.4	462.9	19.8	865.6	854.5	835.9	789.8	748.3	885.5	1016.6	879.6	887.1	1116.9	1034.0	1204.3	1356.8
18	400P154	18	2470.4	2060.8	2323.5	2226.2	2212.0	2207.2	1731.7	1983.7	1963.0	679.8	681.8	681.4	720.0	724.1	599.3	442.4	431.5	0.0	110.3	99.9	129.4	426.1	872.4	861.3	842.7	796.6	755.1	892.3	1023.4	886.4	893.9	1123.7	1040.8	1211.1	1363.6
19	400P158	19	2391.1	2073.1	2335.8	2238.5	2224.3	2219.5	1744.0	1996.0	1975.3	692.1	694.1	693.7	732.3	736.4	611.6	454.7	443.8	110.3	0.0	47.9	77.4	426.1	872.4	861.3	842.7	796.6	755.1	892.3	1023.4	886.4	893.9	1123.7	1040.8	1211.1	1363.6
20	400P159	20	2380.7	2062.7	2325.4	2228.1	2213.9	2209.1	1733.6	1985.6	1964.9	681.7	683.7	683.3	721.9	726.0	601.2	444.3	433.4	99.9	47.9	0.0	49.7	428.0	874.3	863.2	844.6	798.5	757.0	894.2	1025.3	888.3	895.8	1125.6	1042.7	1213.0	1365.5
21	400TP1	21	2410.2	2092.2	2354.9	2257.6	2243.4	2238.6	1763.1	2015.1	1994.4	711.2	713.2	712.8	751.4	755.5	630.7	473.8	462.9	129.4	77.4	49.7	0.0	457.5	903.8	892.7	874.1	828.0	786.5	923.7	1054.8	917.8	925.3	1155.1	1072.2	1242.5	1395.0
22	400TP2	22	2366.6	2048.6	2311.3	2214.0	2199.8	2195.0	1719.5	1971.5	1950.8	667.6	669.6	669.2	707.8	711.9	587.1	7.7	19.8	426.1	426.1	428.0	457.5	0.0	860.2	849.1	830.5	784.4	742.9	880.1	1011.2	874.2	881.7	1111.5	1028.6	1198.9	1351.4
23	425P139	23	2306.6	1988.6	2605.7	2508.4	2494.2	2489.4	2013.9	2265.9	2245.2	962.0	964.0	963.6	1002.2	1006.3	881.5	876.5	865.6	872.4	872.4	874.3	903.8	860.2	0.0	24.9	35.1	113.6	181.7	19.9	951.2	814.2	821.7	1051.5	968.6	1138.9	1291.4
24	425P140	24	2295.5	1977.5	2594.6	2497.3	2483.1	2478.3	2002.8	2254.8	2234.1	950.9	952.9	952.5	991.1	995.2	870.4	865.4	854.5	861.3	861.3	863.2	892.7	849.1	24.9	0.0	24.0	102.5	170.6	44.8	940.1	803.1	810.6	1040.4	957.5	1127.8	1280.3
25	425P141	25	2276.9	1958.9	2576.0	2478.7	2464.5	2459.7	1984.2	2236.2	2215.5	932.3	934.3	933.9	972.5	976.6	851.8	846.8	835.9	842.7	842.7	844.6	874.1	830.5	35.1	24.0	0.0	83.9	152.0	55.0	921.5	784.5	792.0	1021.8	938.9	1109.2	1261.7
26	425P145	26	2230.8	1912.8	2529.9	2432.6	2418.4	2413.6	1938.1	2190.1	2169.4	886.2	888.2	887.8	926.4	930.5	805.7	800.7	789.8	796.6	796.6	798.5	828.0	784.4	113.6	102.5	83.9	0.0	105.9	133.5	875.4	738.4	745.9	975.7	892.8	1063.1	1215.6
27	425P149	27	1906.5	1871.3	2488.4	2391.1	2376.9	2372.1	1896.6	2148.6	2127.9	844.7	846.7	846.3	884.9	889.0	764.2	759.2	748.3	755.1	755.1	757.0	786.5	742.9	181.7	170.6	152.0	105.9	0.0	201.6	833.9	696.9	704.4	934.2	851.3	1021.6	1174.1
28	425TP2	28	2326.5	2008.5	2625.6	2528.3	2514.1	2509.3	2033.8	2285.8	2265.1	981.9	983.9	983.5	1022.1	1026.2	901.4	896.4	885.5	892.3	892.3	894.2	923.7	880.1	19.9	44.8	55.0	133.5	201.6	0.0	971.1	834.1	841.6	1071.4	988.5	1158.8	1311.3
29	450PP137S	29	2138.2	1820.2	2756.7	2659.4	2645.2	2640.4	2164.9	2416.9	2396.2	1113.0	1115.0	1114.6	1153.2	1157.3	1032.5	1027.5	1016.6	1023.4	1023.4	1025.3	1054.8	1011.2	951.2	940.1	921.5	875.4	833.9	971.1	0.0	298.2	598.9	883.1	800.2	970.5	1123.0
30	450PP150S	30	2001.2	1683.2	2619.7	2522.4	2508.2	2503.4	2027.9	2279.9	2259.2	976.0	978.0	977.6	1016.2	1020.3	895.5	890.5	879.6	886.4	886.4	888.3	917.8	874.2	814.2	803.1	784.5	738.4	696.9	834.1	298.2	0.0	461.9	746.1	663.2	833.5	986.0
31	450TP1	31	2008.7	1690.7	2627.3	2530.0	2515.7	2510.9	2035.4	2287.4	2266.7	983.5	985.5	985.1	1023.7	1027.8	903.0	898.0	887.1	893.9	893.9	895.8	925.3	881.7	821.7	810.6	792.0	745.9	704.4	841.6	598.9	461.9	0.0	753.6	670.7	841.0	993.5
32	475P131	32	1884.9	1566.9	2857.0	2759.7	2745.5	2740.7	2265.2	2517.2	2496.5	1213.3	1215.3	1214.9	1253.5	1257.6	1132.8	1127.8	1116.9	1123.7	1123.7	1125.6	1155.1	1111.5	1051.5	1040.4	1021.8	975.7	934.2	1071.4	883.1	746.1	753.6	0.0	153.3	637.6	869.7
33	475PP137S	33	1802.0	1484.0	2774.1	2676.8	2662.6	2657.8	2182.3	2434.3	2413.6	1130.4	1132.4	1132.0	1170.6	1174.7	1049.9	1044.9	1034.0	1040.8	1040.8	1042.7	1072.2	1028.6	968.6	957.5	938.9	892.8	851.3	988.5	800.2	663.2	670.7	153.3	0.0	554.7	786.8
34	475TP1	34	1972.3	1654.3	2944.4	2847.1	2832.9	2828.1	2352.6	2604.6	2583.9	1300.7	1302.7	1302.3	1340.9	1345.0	1220.2	1215.2	1204.3	1211.1	1211.1	1213.0	1242.5	1198.9	1138.9	1127.8	1109.2	1063.1	1021.6	1158.8	970.5	833.5	841.0	637.6	554.7	0.0	957.1
35	500TP1	35	1734.8	1416.8	3096.9	2999.6	2985.4	2980.6	2505.1	2757.1	2736.4	1453.2	1455.2	1454.8	1493.4	1497.5	1372.7	1367.7	1356.8	1363.6	1363.6	1365.5	1395.0	1351.4	1291.4	1280.3	1261.7	1215.6	1174.1	1311.3	1123.0	986.0	993.5	869.7	786.8	957.1	0.0

The name of the dump and their capacity.

Dump name	Dump name in the Software	Dump capacity (ton)
A	dump1	40000
B	dump2	40000
C	dump3	40000
D	dump4	40000
E	dump5	40000
Waste Dump (surface/louhe)	dump6	1000000000
Waste Dump (surface, PAF/513K)	dump7	1000000000

The development fleet used in the software, with moving speed an operating speed.

Iden-tifica-tion used in the soft-ware	Devel-opment fleet	Ma-chine model	Equip ment amou nt	Aver-age time per 1 round (4.7 m) in hour	Note for the moving speed	Operat-ing speed (me-ter/hour)	Moving speed (km/ho ur)	Moving speed (me-ter/hou r)	Data source
Ma-chine1	Scaling	Normet Scame c	2	2	maximum value taken	2.35	7.00	7000	http://www.nor-met.com/in et/normet/p rod-ucts.nsf/(At tach-mentsByFil eName)/28 547412.pdf /$File/2854 7412.pdf
Ma-chine2	Cleaning excava-tor+loade r com-bined	Exca-va-tor=Me calec; load-er=CA T 930	3 (2 exca-va-tor+1 loader)	1	average of exca-vator and loader speed, where excava-tor=assumed 80% of max moving speed, load-er=average speed between 2nd and 3rd gear forward	4.70	19.47	19467	http://www.nor-met.com/in et/normet/fl ow.nsf/(Att ach-mentsByFil eName)/Spr ay-mec%2010 50%20WP C%201800 045.pdf/$Fi le/Sprayme c%201050 %20WPC% 201800045. pdf
Ma-chine3	Shotcret-er	Normet Spray mec 1050	2	1	for curing time. Speed assumed 2nd gear	4.70	10.00	10000	http://www.atlascop-co.us/image s/technical %20specifi cati-on%20bolte c%20lc_98 51%202201 %2001g_tc m795-1532552.pd f
Ma-chine4	Support - Bolting	Atlas Copco Boltec LC	3	3	average between inclined and flat ground speed taken	1.57	10.00	10000	http://www.mecalac.co m/en/machi ne/10msx.h tml
Ma-chine5	Drilling	Atlas Copco Rocket Boom-er E2 C22	2	3	average between inclined and flat ground speed taken	1.57	10.00	10000	http://www.atlascop-co.com/ima ges/technic al%20speci fica-

									tion%20boomer%20e2%20c_9851%202451%2001f_web_tcm834-1533272.pdf
Machine6	Charging	Normet Charmec 1610	2	1.5	average between inclined (1:10) and flat ground speed taken	3.13	16.50	16500	http://www.kellytractor.com/eng/images/pdf/earthmoving/wheel_loaders/930g.pdf
Machine7	Services - Water truck	Scania	1	1	maximum value taken	4.70	18.00	18000	Agnico Eagle Kittilä

The information about the truck travel speed, loading time of the truck and its payload capacity.

Truck ID	**Truck speed *(meter/hour)***	**Loading time of the truck *(hour)***	**Truck payload *(ton)***
1	25000	0.083	27
2	25000	0.083	27
3	25000	0.083	27
4	25000	0.083	27
5	25000	0.083	27
6	25000	0.083	27
7	25000	0.083	27
8	25000	0.083	27
9	25000	0.083	27
10	25000	0.083	27

The machine set for each machine type and their availability status.

Fleet number	**Machine type *(anything per fleet)***	**Availability of the machine, 0 means not available, 1 means available**
1	scaler	1
1	cleaning	1
1	shotcreter	1
1	bolter	1
1	driller	1
1	charger	1
1	dustsprayer	1
2	scaler	1
2	cleaning	1
2	shotcreter	1
2	bolter	1
2	driller	1
2	charger	1
2	dustsprayer	0
3	scaler	0
3	cleaning	0
3	shotcreter	0
3	bolter	1
3	driller	0
3	charger	0
3	dustsprayer	0

The dependency status of one workface to another workface.

Workface ID	The workface where this ID is dependent on (if it's not dependent on any ID, fill in with the ID of that workface itself)
1	1
2	2
3	3
4	4
5	6
6	6
7	7
8	8
9	9
10	10
11	11
12	12
13	13
14	14
15	15
16	17
17	16
18	18
19	19
20	20
21	21
22	22
23	24
24	25
25	26
26	27
27	27
28	28
29	29
30	30
31	31
32	32
33	31
34	34
35	34

The amount of blasted material that has to be removed from the workface.

Workface_ID	Blasted material at workface *(ton)*
1	450
2	450
3	8500
4	8500
5	8500
6	450
7	8500
8	8500
9	8500
10	8500
11	8500
12	8500
13	8500

14	450
15	450
16	8500
17	8500
18	8500
19	8500
20	8500
21	450
22	450
23	8500
24	8500
25	8500
26	8500
27	8500
28	450
29	8500
30	8500
31	450
32	8500
33	8500
34	450
35	450

Workface priority according to the setup level.

	Working face_ID	**Priority level value of the routing face**
VT-TUT	1	1
VT1	2	1
200P134	3	1
200P140	4	1
225P122	5	1
225TP2	6	1
250P137	7	1
250P122N	8	1
250PP120S	9	1
375P153	10	1
375P154	11	1
375P155	12	1
375P159	13	1
375TP1	14	1
375TP2	15	1
400P135	16	1
400P136	17	1
400P154	18	1
400P158	19	1
400P159	20	1
400TP1	21	1
400TP2	22	1
425P139	23	1
425P140	24	1
425P141	25	1
425P145	26	1
425P149	27	1
425TP2	28	1
450PP137S	29	1

450PP150S	30	1
450TP1	31	1
475P131	32	1
475PP137S	33	1
475TP1	34	1
500TP1	35	1

The workload that has to be done for each machine type in each workface.

	Workload per machine type at each workface								
Work face_ X	Workface name	Total round needed for drilling	Total workload per machine type						
			Ma-chine type 1	Ma-chine type 2	Ma-chine type 3	Ma-chine type 4	Ma-chine type 5	Ma-chine type 6	Ma-chine type 7
1	VT-TUT	2	9	9	9	9	9	9	9
2	VT1	2	9	9	9	9	9	9	9
3	200P134	2	9	9	9	9	9	9	9
4	200P140	2	9	9	9	9	9	9	9
5	225P122	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
6	225TP2	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
7	250P137	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
8	250P122N	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
9	250PP120S	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
10	375P153	Fan drilling	3	3	3	3	3	3	3
11	375P154	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
12	375P155	Fan drilling	3	3	3	3	3	3	3
13	375P159	Fan drilling	3	3	3	3	3	3	3
14	375TP1	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
15	375TP2	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
16	400P135	Fan drilling	3	3	3	3	3	3	3
17	400P136	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
18	400P154	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
19	400P158	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
20	400P159	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
21	400TP1	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
22	400TP2	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
23	425P139	Fan drilling	3	3	3	3	3	3	3
24	425P140	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
25	425P141	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
26	425P145	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
27	425P149	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
28	425TP2	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
29	450PP137S	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
30	450PP150S	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
31	450TP1	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
32	475P131	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
33	475PP137S	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
34	475TP1	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5
35	500TP1	1	4.5	4.5	4.5	4.5	4.5	4.5	4.5

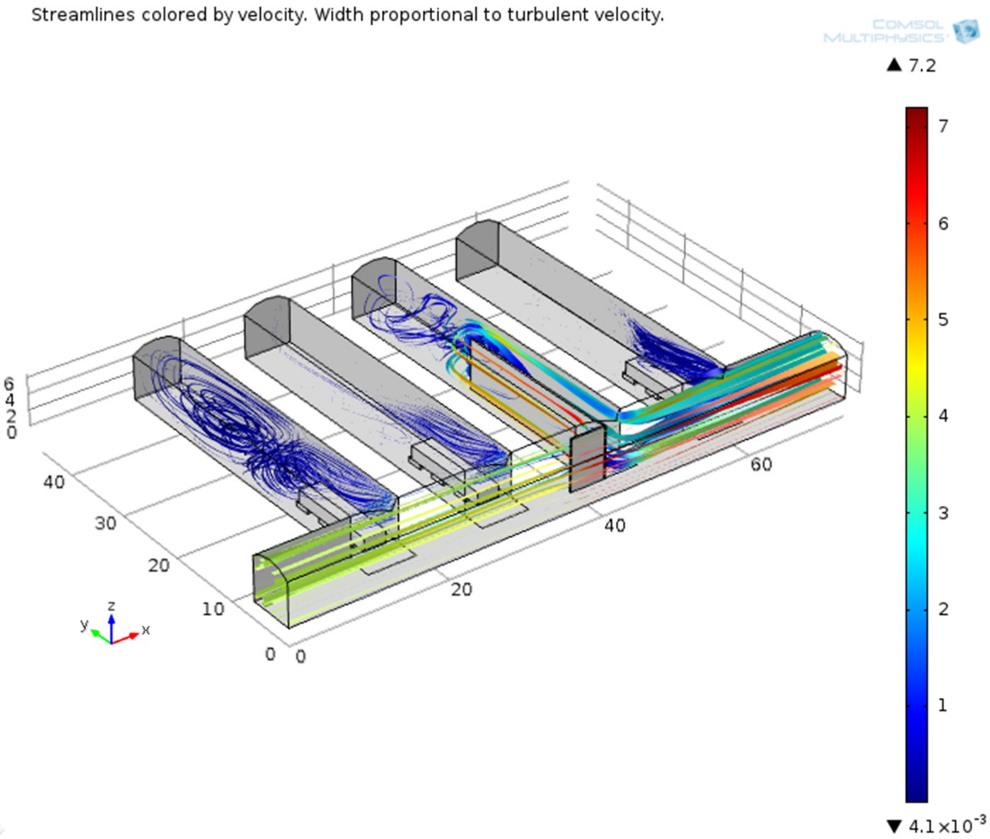
Distance from each workface to each dump.

Distance from Workface_x to Dump_y			**Dump name**						
			A	**B**	**C**	**D**	**E**	**Waste Dump (sur-face/louhe)**	**Waste Dump (surface, PAF/513 K)**
Num-ber	**Work-face/Drift name**	**Work-face ID**	**Dump 1**	**Dump 2**	**Dump 3**	**Dump 4**	**Dump 5**	**Dump 6**	**Dump 7**
1	VT-TUT	1	5410.7	5410.7	5410.7	5410.7	5410.7	5820.7	5820.7
2	VT1	2	5092.7	5092.7	5092.7	5092.7	5092.7	5502.7	5502.7
3	200P134	3	2725.6	2725.6	2725.6	2725.6	2725.6	3135.6	3135.6
4	200P140	4	2628.3	2628.3	2628.3	2628.3	2628.3	3038.3	3038.3
5	225P122	5	2892.1	2892.1	2892.1	2892.1	2892.1	3177.3	3177.3
6	225TP2	6	2887.3	2887.3	2887.3	2887.3	2887.3	3297.3	3297.3
7	250P137	7	2684.8	2684.8	2684.8	2684.8	2684.8	3094.8	3094.8
8	250P122N	8	2936.8	2936.8	2936.8	2936.8	2936.8	3346.8	3346.8
9	250PP120S	9	2916.1	2916.1	2916.1	2916.1	2916.1	3326.1	3326.1
10	375P153	10	3331.5	3331.5	3331.5	3331.5	3331.5	3741.5	3741.5
11	375P154	11	3333.5	3333.5	3333.5	3333.5	3333.5	3743.5	3743.5
12	375P155	12	3333.1	3333.1	3333.1	3333.1	3333.1	3743.1	3743.1
13	375P159	13	3371.7	3371.7	3371.7	3371.7	3371.7	3781.7	3781.7
14	375TP1	14	3375.8	3375.8	3375.8	3375.8	3375.8	3785.8	3785.8
15	375TP2	15	3251.0	3251.0	3251.0	3251.0	3251.0	3661.0	3661.0
16	400P135	16	3626.2	3626.2	3626.2	3626.2	3626.2	4036.2	4036.2
17	400P136	17	3615.3	3615.3	3615.3	3615.3	3615.3	4025.3	4025.3
18	400P154	18	3622.1	3622.1	3622.1	3622.1	3622.1	4032.1	4032.1
19	400P158	19	3634.4	3634.4	3634.4	3634.4	3634.4	4044.4	4044.4
20	400P159	20	3371.7	3371.7	3371.7	3371.7	3371.7	4034.0	4034.0
21	400TP1	21	3653.5	3653.5	3653.5	3653.5	3653.5	4063.5	4063.5
22	400TP2	22	3609.9	3609.9	3609.9	3609.9	3609.9	4019.9	4019.9
23	425P139	23	3904.3	3904.3	3904.3	3904.3	3904.3	4314.3	4314.3
24	425P140	24	3893.2	3893.2	3893.2	3893.2	3893.2	4303.2	4303.2
25	425P141	25	3874.6	3874.6	3874.6	3874.6	3874.6	4284.6	4284.6
26	425P145	26	3828.5	3828.5	3828.5	3828.5	3828.5	4238.5	4238.5
27	425P149	27	3787.0	3787.0	3787.0	3787.0	3787.0	4197.0	4197.0
28	425TP2	28	3924.2	3904.3	3904.3	3904.3	3904.3	4334.2	4334.2
29	450PP137S	29	4055.3	4055.3	4055.3	4055.3	4055.3	4465.3	4465.3
30	450PP150S	30	3918.3	3918.3	3918.3	3918.3	3918.3	4328.3	4328.3
31	450TP1	31	3925.8	3925.8	3925.8	3925.8	3925.8	4335.8	4335.8
32	475P131	32	4155.6	4155.6	4155.6	4155.6	4155.6	4565.6	4565.6
33	475PP137S	33	4072.7	4072.7	4072.7	4072.7	4072.7	4482.7	4482.7
34	475TP1	34	4243.0	4243.0	4243.0	4243.0	4243.0	4653.0	4653.0
35	500TP1	35	4395.5	4395.5	4395.5	4395.5	4395.5	4805.5	4805.5

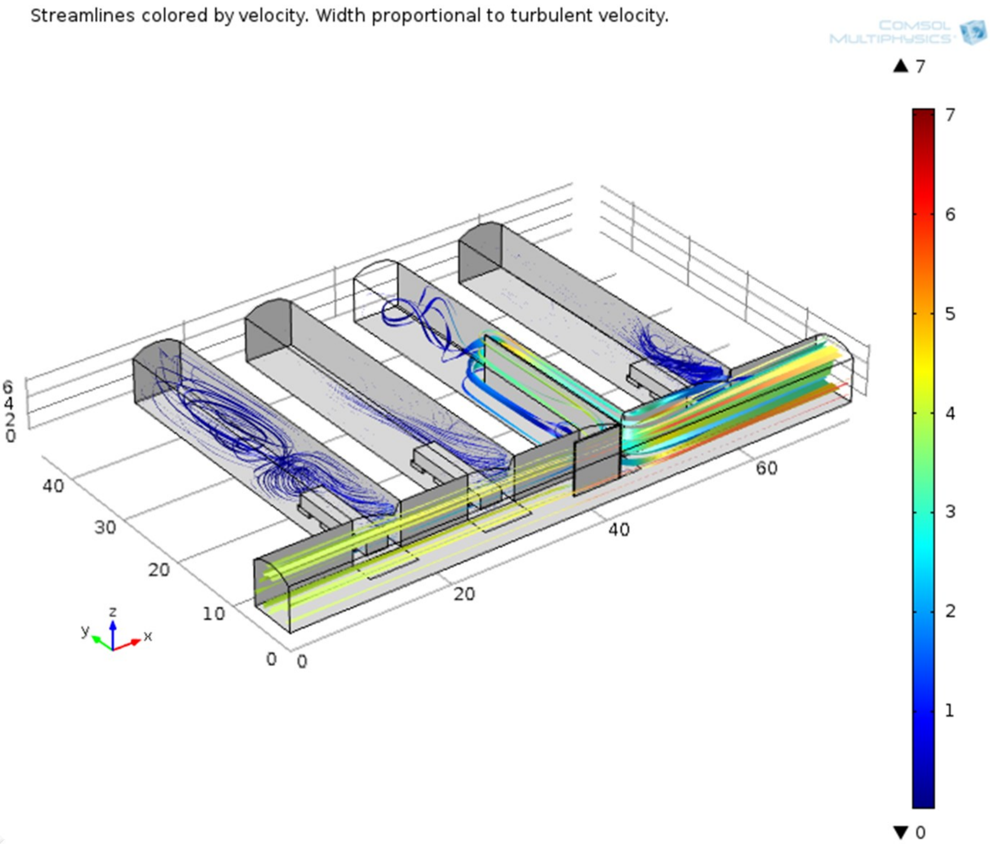
Appendix 2. Example of ventilation cost together with other costs according to InfoMine USA (2010)

Daily Ore Production (tonnes)	200	1,000	2,000
Equipment Costs (dollars/unit)			
Production Driils	$ 709,000	$ 709,000	$ 709,000
Production Scoop Trams	455,500	800,000	925,000
Vertical Development Drills	709,000	709,000	709,000
Horizontal Development Drills	455,500	800,000	925,000
Development Scoop Trams	3,821,400	4,253,700	5,237,100
Production Hoists	1,219,900	1,269,200	1,444,200
Rock Bolters	766,000	766,000	766,000
Shotcreters	35,600	63,400	63,400
Drain Pumps	14,200	20,800	24,100
Fresh Water Pumps	7,340	7,340	7,340
Backfill Mixers	11,700	11,800	11,800
Backfill Pumps	29,900	60,000	60,000
Service Vehicles	276,300	281,000	296,100
Compressors	44,200	44,200	44,200
Ventilation Fans	111,000	171,800	243,900
Exploration Drills	73,500	73,500	73,500
COST SUMMARY			
Operating Costs (dollars/tonne ore)			
Equipment Operation	$ 6.74	$ 5.24	$ 5.62
Supplies	18.43	16.93	15.64
Hourly Labor	30.80	15.74	12.47
Administration	19.24	9,80	6.73
Sundries	7.52	4.77	4.05
Total Operating Costs	**$ 82.73**	**52.48**	**44.51**
Unit Operating Cost Distribution (dollars/tonne ore)			
Drifts/Ramps	6	3.78	1.94
Crosscuts	7.11	4.57	2.95
Ore Passes	0.23	0.37	0.36
Ventilation Raises	0.45	0.47	0.45
Main Haulage	5.89	3.34	2.73
Backfill	6.79	6.83	7.02
Services	16.16	8.40	7.58
Ventilation	0.96	0.57	0.87
Exploration	2.24	1.37	0.87
Maintenance	1.77	0.72	0.67
Administration	15.40	5.84	4.46
Miscellaneous	7.52	4.77	4.05
Total Operating Costs	$ 83	$ 52	$ 45
Capital Costs			
Equipment Purchase	$ 13,731,300	$ 36,604,300	$ 50,646,200
Preproduction Underground Excavation			
Shafts	3,808,000	6,349,500	9,615,100
Drifts/Ramps	109,700	1,363,300	2,103,700
Crosscuts	113,200	1,646,000	2,502,500
Ore Passes	61,600	206,800	345,100
Ventilation Raises	351,500	766,800	1,256,600
Surface Facilities	2,671,800	6,419,300	8,710,700
Working Capital	882,700	2,799,700	4,748,500
Engineering & Management	2,710,100	6,936,300	9,773,400
Contingency	2,084,700	5,335,600	7,518,000
Total Capital Costs	**$ 26,524,600**	**$ 68,427,600**	**$ 97,219,800**
Total Capital Cost per Dally Tonne Ore	**$ 132,623**	**$ 68,428**	**$ 48,610**

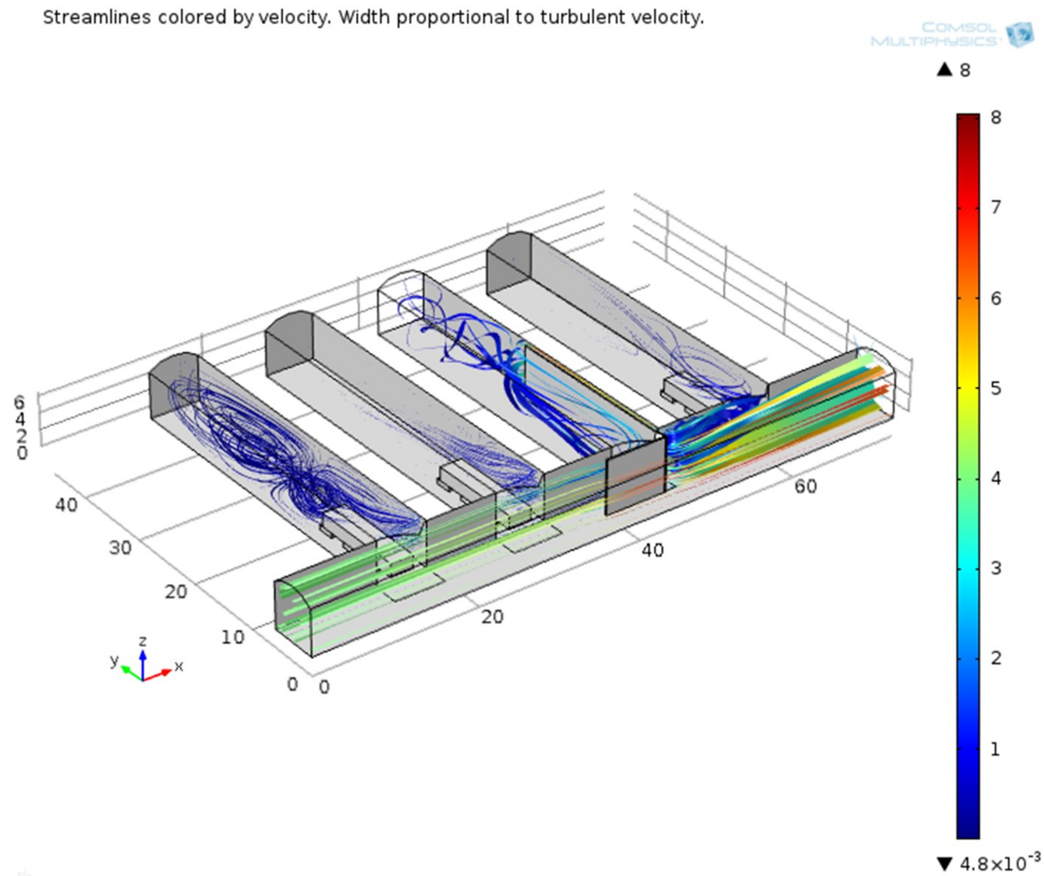
Appendix 3. 3D plotting of the simulation results when using different air brattice configurations



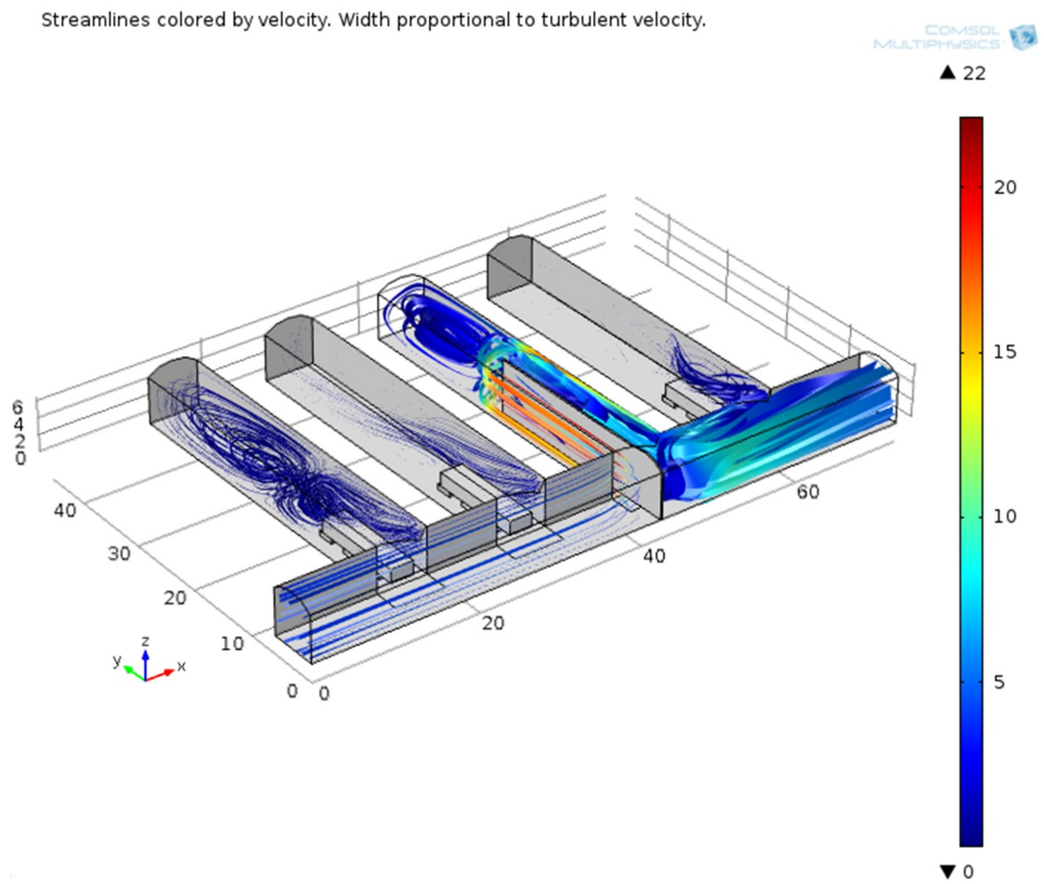
Air brattice configuration 1, L shaped, 1.5 meter to the left side wall.



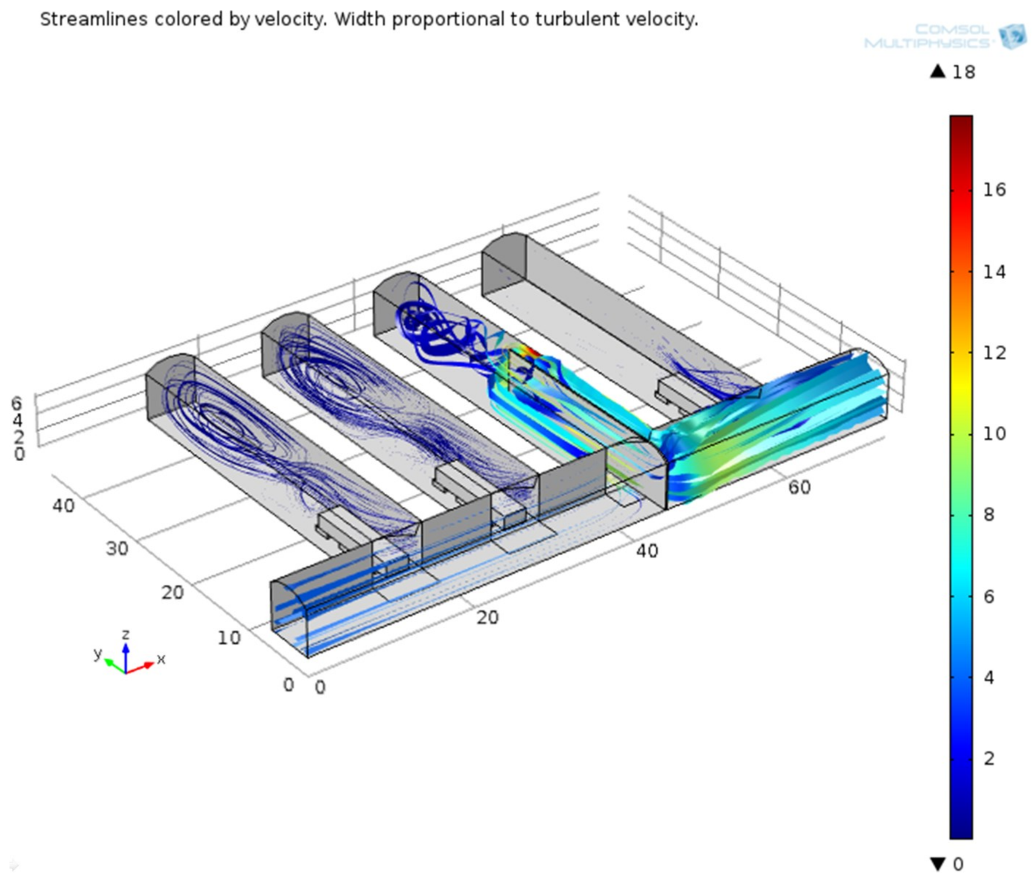
Air brattice configuration 2, L shaped, 3 meter to the left side wall.



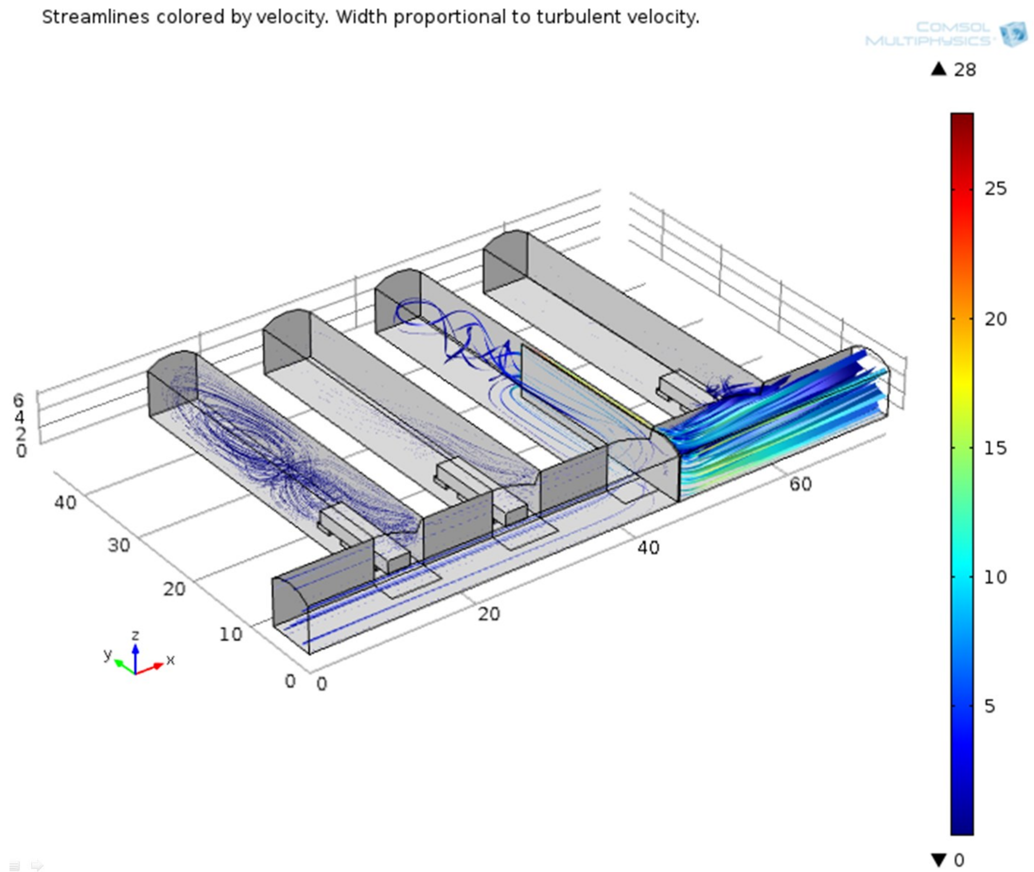
Air brattice configuration 3, L shaped, 4.5 meter to the left side wall.



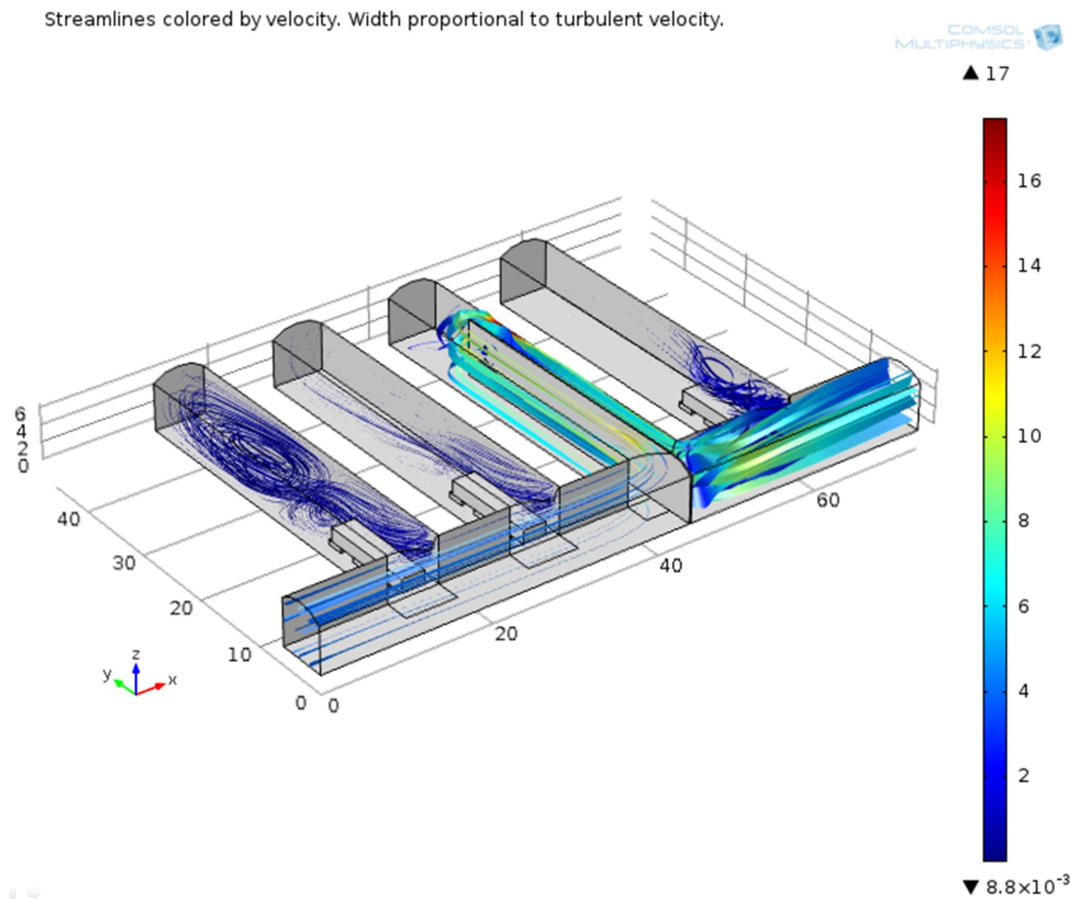
Air brattice configuration 4, straight, 1.5 meter to the left side wall.



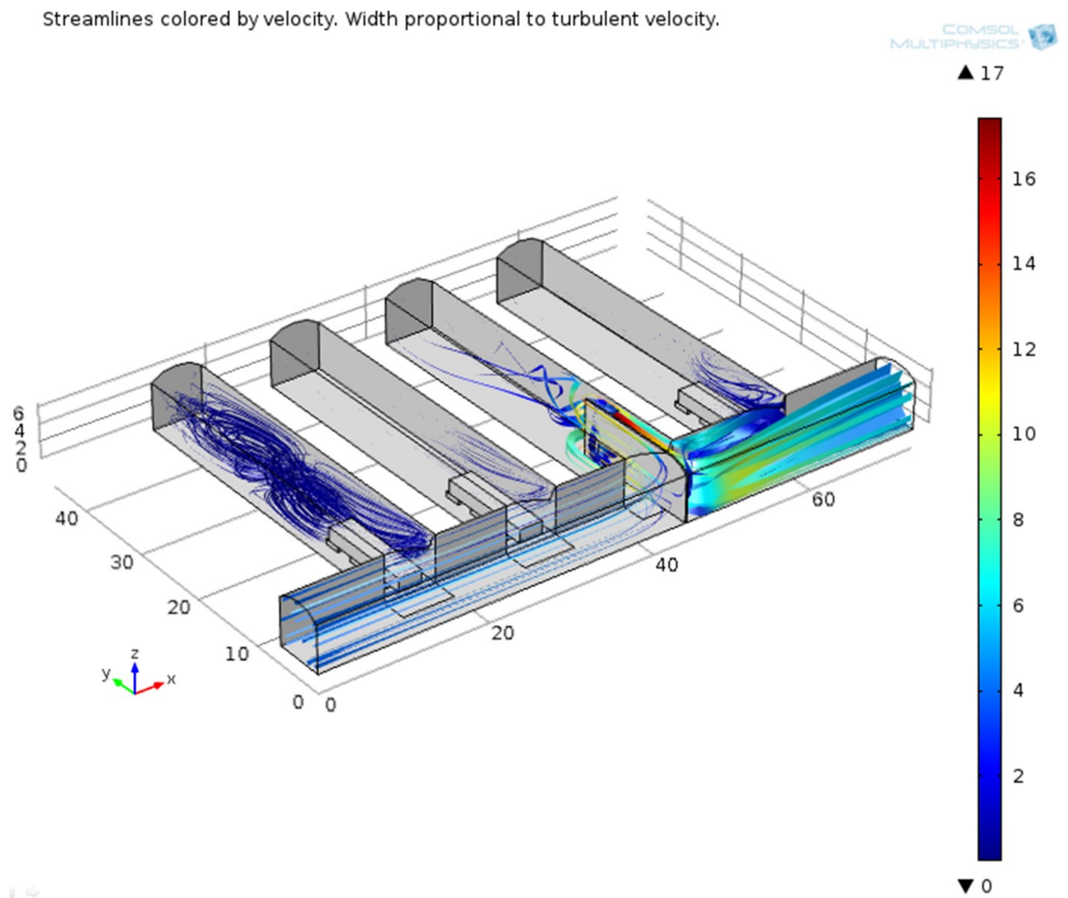
Air brattice configuration 5, straight, 3 meter to the left side wall.



Air brattice configuration 6, straight, 4.5 meter to the left side wall.



Air brattice configuration 7, straight, 3 meter to the left side wall, 30 meters long.



Air brattice configuration 8, straight, 4.5 meter to the left side wall.